noranda

CONTRACT REPORT NO. 0SQ84-00231

'A PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT'

DSS FILE NO. 07SQ.23440-4-9071

MILESTONE 1 REPORT
MINE SITE SELECTION

SUBMITTED BY:

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MICROMEDIA

135pp

DECLASSIFIED
DATE 18/7/86
AUTHORIZED BY

EXECUTIVE SUMMARY

Milestone 1, the Mine Site Selection, has been attained. Seven Noranda mines sites were visited and considered for in-situ bacterial leaching tests. The Geco mine was selected as the recommended site on which to conduct pre-feasibility studies. The Mattabi and Heath Steele mines were identified as alternative sites.

The attached report contains site reports for each of the mines visited, and discussion of factors considered in evaluating each site.

The overall project is on schedule and within budget. Work will begin immediately on Tasks 2 and 3

SOMMAIRE

L'étape 1, soit l'Identification des Sites Miniers, a été exécutée. Sept propriétés minières de la compagnie Noranda fûrent visitées et examinées quant à la possibilité d'utiliser la lixiviation bactériologique. La Mine Geco a été retenue comme site plausible pour conduire une étude de pré-rentabilité. Les propriétés Mattabi et Heath Steele fûrent retenues comme sites de rechange.

La rapport ci-joint contient une évaluation de chaque site visité et une analyse des facteurs considerés.

Le projet se déroule selon l'échéancier et en deçà du budget alloué. Le travail des étapes 2 et 3 à déjà débuté.

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(Pamour Porcupine)

H List of Minerals Referred to in this. 108
Report, with their formulae

PART I

a) The project is on schedule, as illustrated in the ''Plan of Work and Cost'' included in Part III.

b) The project is within budget.

c) The project is free of any areas of technical concern on which Crown guidance is required.

A problem exists with the present time allowance for Tasks 2, 3 and 4. Outside consulting involvement will be required in completing tests for Tasks 2 and 3. The logistics involved make January 31, 1985 an unrealistic date for presentation of Milestone 2 and 3 reports.

PART II

a) Task 1, the Mine Site Selection, has been completed as required. Work will begin immediately on Tasks 2 and 3. The project as a whole is progressing satisfactorily.

b) No variation from the plan of work was made during the completion of Task 1.

c) No major items of equipment were purchased during the completion of Task 1.

d) Seven Noranda mine sites were visited and considered as sites for in-situ bacterial leaching tests. The mines visited were:

Mines Gaspé, Quebec.
Heath Steele Mine, New Brunswick
Remnor Project, Quebec
Goldstream Mine, British Columbia
Geco Mine, Ontario
Mattabi Mine, Ontario
Copper Mine, Pamour Porcupine Mines Ltd., Ontario

The locations of these mines are illustrated in Figure 1.

Management at all of the mines was very receptive to the possible testing of bacterial leaching.

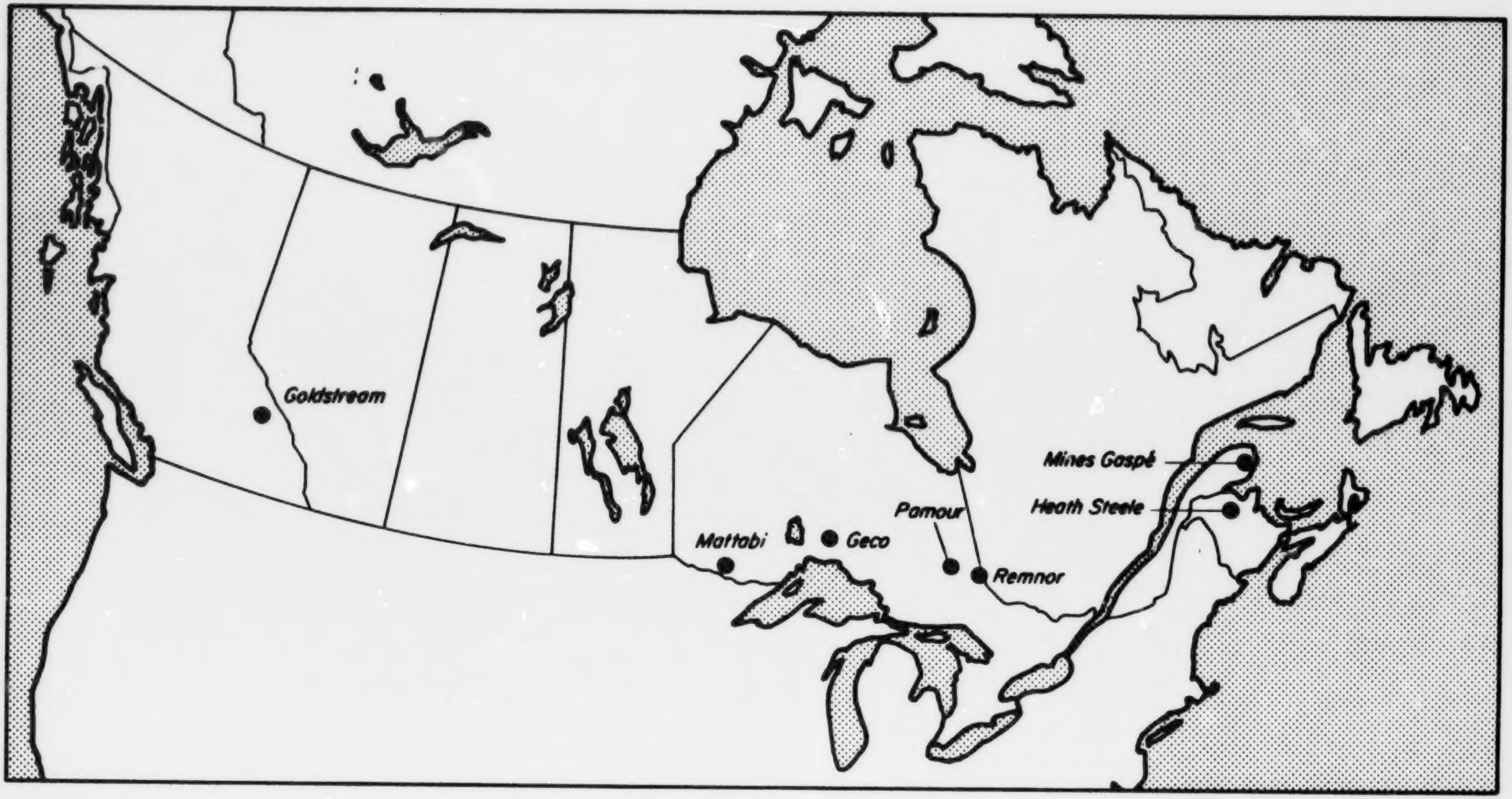
Goldstream
Mattabi
Geco
Pamour
Remnor
Mines Gaspé
Heath Steele

Figure 1. Locations of Noranda Mine Sites Visited.

The criteria used to evaluate each mine are listed in Table I. The maximum weight assigned to each factor indicates its relative potential for positive or negative influence on the success of in-situ leaching. These weights reflect Noranda's set of priorities. Another company conducting the same exercise may use the same criteria, but attach entirely different maximum weights to them. The evaluation process is, of necessity, qualitative at this stage. A quantitative set of criteria will be possible only after completion of the pre-feasibility study.

DESCRIPTIONS OF MINE SITE SELECTION CRITERIA

-General Description of Mine Site

The mine's location, surrounding topography and climatology are considered to have little effect in most cases on its potential as a test site. However, when a mine is located in mountainous terrain and above the water table, the potential for contamination of local rivers and lakes becomes an important consideration. Climatology may have an influence where test zones are below an open pit, and therefore affected by air temperature; or where there is a correlation between precipitation and mine water pumping volumes. These situations would all lead to a negative rating of the mine. The choice between a -1 rating and a -2 rating (for topography for example) would depend on the relative importance of topography at the particular mine site in question.

-Mine Configuration (General)

The mining method(s) in use at the mine and the tonnage handled were reported for information purposes, but do not influence the evaluation. A maximum weight of 3 is assigned to the orebody dimensions and dip. .

A steeply dipping or vertical orebody would merit a positive rating, as it would be more amenable to leaching than an orebody of shallow dip. A +1 rating for surface facilites and layout would be awarded for a mine where there was ample space for a pilot plant (200 m x 200 m) near the shaft. A +2 rating would be awarded when equipment for leaching (pregnant solution handling/treatment) exists on site, (e.g., vat leaching and cementation plant at Mines Gaspé).

TABLE I

MINE SITE SELECTION CRITERIA

	MAXIMUM WEIGHT
GENERAL DESCRIPTION OF MINE SITE	
- Location	0
- Topography	2
- Climatology	1
MINE CONFIGURATION (General)	
- Orebody Dimensions and Dip	3
- Mining Method(s) Used	0
- Tonnage Handled	0
- Surface Facilities and Layout	2
GEOLOGY	
- Mineralogy, Chemical Analysis	3
- Structure	4
MINE WATER	
- Volume Handled	2
- pH	4
VENTILATION	
- Volume Available	1
POTENTIAL ZONES FOR BACTERIALLY-ASSISTED LEACHING	
- Tonnage	2
- Grade of Copper	3
- Grade of Zinc	2
- Grades of Precious Metals	3
- Ground Temperature	1
SITE SPECIFIC FACTORS	
- For example: attitude of mine management, potential for damage to the environment, future prospects for mine using conventional techniques, access to potential test zones.	1 to 4 for each factor

-Geology

The mineralogy and chemical composition of the material to be leached is assigned a maximum weight of 3. The presence of acid consuming gangue in ore or waste zones would warrant a negative rating. The less complex the mineralogy of a potential test zone, the more strongly positive the rating for this factor would be.

Geological structure of the test zone, and the mine in general, is weighted very heavily (maximum weight = 4) because the ability to contain and control a leaching solution will be dependent on the number, spacing and nature of fractures, joints and faults in the rock mass. The more fractured is the rock surrounding a potential test zone, the more negative the rating. If structural mapping has been done, data for 'fractures per metre' may be available. The RQD, or Rock Quality Designation, may also be used. The RQD is defined as the sum length of all pieces of core greater than or equal to two times the core diameter divided by the total length of core recovered from a diamond drill hole. RQD is usually expressed as a percentage. Where no structural information is available, a rating must be based on visual observation and the judgment of the mine's gangue.

-Mine Water

The volume of mine water handled is an important consideration only if there is a correlation between daily volume pumped and precipitation. This may be the case where the zone is close to surface and the rock mass is of moderate to high permeability. In such a case (e.g., Mines Gaspé), a negative rating would be assigned to reflect the potential for migration of the leaching solution into the local groundwater system.

Mine water pH is regarded as a very important factor (maximum weight = 4). A positive rating is assigned where the pH is acidic, since leaching operation would not disrupt existing pumping networks and indicates that the gangue is not acid-consuming.

-Ventilation

The volume of ventilation available to the potential test zone(s) is recorded for information purposes. At this stage it is not possible to determine accurately the amount required for leaching reactions although it is expected that no additional ventilation over normal operation would be required. Therefore, this factor is not heavily weighted.

-Potential Zones for Bacterially-Assisted Leaching

To ensure reasonable production rates of copper from low grade zones, it will be necessary to have large amounts of leaching material. Therefore, a positive rating is assigned where large tonnages are available.

The grade factor (for copper, zinc and precious metals) are interrelated, since Canadian sulphide deposits are generally polymetallic in nature. The higher the copper grade, the more positive the rating. The precious metals grade factor is assigned a maximum weighting of 3 (the same as copper) because leaching will not allow recovery of gold or silver. Therefore, the higher the grades of these metals, the more negative the rating will be. Zinc will be leached, but previous internal studies have shown that its recovery from dilute solution is uneconomic. For practical purposes, higher zinc grades are considered detrimental and result in a negative rating.

The ground temperature is not felt to have a significant effect on leaching potential since the solution could be externally heated if required. In addition, some recently reported tests have shown that bacterially-assisted leaching might be effected at lower temperatures. However, positive ratings were assigned in the cases of Geco and Goldstream, which have high temperatures relative to the other mines visited.

-Site Specific Factors

Any given mine might have one or more factors which substantially affect or even outweigh the combined influence of all the other factors. The maximum weight of a given site specific factor depends on its relative importance at that particular mine.

-Mine Site Evaluations

Appendices A through G contain site reports for each of the mines visited, while Appendix H contains a list of the minerals referred to in the reports, with their formulae.

Table II contains a summary of the evaluations conducted for each mine. and Table III summarizes the main advantages and disadvantages of each site.

All of the mines considered share the advantage of ready access to potential test zones by existing development; and the disadvantage of low ambient rock temperatures. The Geco mine has the greatest potential for in-situ tests, primarily because of relatively high ambient rock temperatures, a steeply dipping orebody and a potential test zone well away from current mining areas.

Two alternative sites, the Mattabi mine and the Heath Steele mine (in order of preference) have been identified as back-ups to be reconsidered if Geco should prove unsuitable.

Site specific factors affected the overall ratings for two mines. The Remnor Project is a gold mine with copper as an important secondary mineral. Potential test zones for copper sulphide leaching contain gold that is included in Remnor's reserves. The zones are also small and widely scattered within the mine, which would cause complicated pumping layouts and extensive development. These detrimental factors combine for a -4 rating.

The Goldstream deposit has high grades of copper and zinc relative to potential test zones in the other six mines. Goldstream has reasonable prospects for being reopened using conventional methods under improved market conditions. A rating of -4 is assigned to reflect this factor.

TABLE II

SUMMARY OF MINE SITE EVALUATIONS

	MINES GASPE	HEATH STEELE	REMNOR	GOLDSTREAM	GECO	MATTABI	PAMOUR PORCUPINE (COPPER MINE)
General Description of Mine Site							
- Topography	-2	+1	+1	-2	+1	+1	+1
- Climatology	-1	0	0	0	0	0	0
Mine Configuration (General)							
- Orebody Dimensions and Dip	-3	+2	+1	-2	+3	+2	+2
- Surface Facilities and Layout	+2	+1	+1	+1	+1	+1	+1
Geology							
- Mineralogy, Chemical Analysis	+3	-1	+1	-1	+3	+2	-2
- Structure	-3	-1	0	0	+2	+2	-3
Mine Water							
- Volume Handled	-2	+1	+1	0	-1	0	0
- pH	-4	+4	+4	-4	+4	+4	+4
Ventilation							
- Volume Available	+1	+1	+1	+1	+1	+1	+1
Potential Zones for Leaching							
- Tonnage	+2	+1	+2	+2	+2	-1	+1
- Grade of Copper	+3	+2	+1	+3	+1	+1	+1
- Grade of Zinc	0	-1	0	-2	+1	-1	0
- Grade of Precious Metals	0	-1	-2	0	+1	-1	-1
- Ground Temperature	0	-1	0	+1	+1	0	0
Sub-Total	-4	+8	+10	-3	+20	+11	+5
Site Specific Factors (See Text for Explanation)	0	0	-4	-4	0	0	0
TOTAL SCORE	-4	+8	+6	-7	+20	+11	+5

TABLE III

MINE SITES CONSIDERED FOR IN-SITU BACTERIAL LEACHING TESTS

MINE	ADVANTAGES	DISADVANTAGES
Recommended Site		
GECO (Appendix E) (Total Score = +20)	Steeply dipping orebody. Relatively high ambient rock temperature (13°). Potential material for leaching (1000000 t at < 1% Cu) is isolated from current mining areas, and is considered 'waste'.	Tonnage of potential material for leaching is limited for a long-term operation.
Alternative Sites		
1) Mattabi (Appendix F) (Total Score = +11)	Reasonable grades (1-1.5% Cu). Mine water is acidic (pH between 3.2 and 3.4).	Limited amount of leachable material (375000 t in the Upper Mine). Mainly a zinc mine, and the zinc and silver values would be lost. Blasting in high Cu areas could affect current mining areas.

TABLE III (Continued)

MINE	ADVANTAGES	DISADVANTAGES
2) Heath Steele (Appendix B) (Total Score = +8)	Accessible zone beneath open pit,could be drilled from surface. Reasonable grades (1.46% Cu). Mine water is naturally acidic (pH$\leqslant$ 4).	Polymetallic massive sulphide ore, silver values would be lost. Low ambient rock temperatures, (5 - 8 °C). Blasting of crown pillar would affect the mine ventilation system and create problems with spring runoff entering the mine.
SITES REJECTED (Order not significant)		
Mines Gaspé (Appendix A) (Total Score = -4)	Existing vat leaching equipment could possibly be used.	All potential material for leaching is in pillars the removal of which could endanger mine stability. Rock mass is quite permeable. Mine water is basic (pH>7).
Remnor (Appendix C) (Total Score = +6)	Mine water is acidic (pH = 3)	Small pockets of ore for leaching. Low grades (about 0.5% Cu) Loss of gold values (>4.5 g/t Au)

TABLE III (Continued)

MINE	ADVANTAGES	DISADVANTAGES
Goldstream (Appendix D) (Total Score = -7)	Reasonably high ambient ground temperature (10-15°C) relative to other mines.	Copper grades high enough (about 3.5%) for possible re-opening of the mine in the foreseeable future. Fairly flat orebody (dip = 33°). Mine water is basic (pH = 7.9). Carbonate in the ore may be disadvantageous for leaching.
Copper Mine (Pamour) (Appendix G) (Total Score = +5)	Existing access to a number of small potential test areas (e.g. stopes containing broken ore).	Low copper grades (0.47% Cu) and loss of gold values (1.37 g/t Au). Ground is fractured and heavily foliated in potential leaching zone, and leach solution may migrate. Presence of carbonate (acid-consuming) is a possible disadvantage.

Appendices A through G contain site reports for each of the mines visited. The main advantages and disadvantages of each site for the purposes of this project are summarized in Table 1. All of the mines considered share the advantage of ready access to potential test zones by existing development; and the disadvantage of low ambient rock temperatures. The Geco mine has the greatest potential for in-situ tests, primarily because of relatively high ambient rock temperatures, a steeply dipping orebody and a potential test zone well away from current mining areas.
Two alternative sites, the Mattabi mine and the Heath Steele mine (in order of preference) have been identified as back-ups to be reconsidered if Geco should prove unsuitable. The other four mine sites were rejected for reasons summarized in Table 1 and discussed in greater detail in the site reports.

Appendix H contains a list of the minerals referred to in the site reports, with their formulae.

TABLE 1

MINE SITES CONSIDERED FOR IN-SITU BACTERIAL LEACHING TESTS

MINE	ADVANTAGES	DISADVANTAGES
Recommended Site		
GECO (Appendix E)	Steeply dipping orebody. Relatively high ambient rock temperature (13°). Potential material for leaching (1000000 t at < 1% Cu) is isolated from current mining areas, and is considered 'waste'.	Tonnage of potential material for leaching is limited for a long-term operation.
Alternative Sites		
1) Mattabi (Appendix F)	Reasonable grades (1-1.5% Cu). Mine water is acidic (pH between 3.2 and 3.4).	Limited amount of leachable material (375000 t in the Upper Mine). Mainly a zinc mine, and the zinc and silver values would be lost. Blasting in high Cu areas could affect current mining areas.

TABLE 1 (Continued)

MINE	ADVANTAGES	DISADVANTAGES
2) Heath Steele (Appendix B)	Accessible zone beneath open pit,could be drilled from surface. Reasonable grades (1.46% Cu). Mine water is naturally acidic ($pH \leq 4$).	Polymetallic massive sulphide ore, silver values would be lost. Low ambient rock temperatures, (5 - 8 °C). Blasting of crown pillar would affect the mine ventilation system and create problems with spring runoff entering the mine.
SITES REJECTED (Order not significant)		
Mines Gaspé (Appendix A)	Existing vat leaching equipment could possibly be used.	All potential material for leaching is in pillars the removal of which could endanger mine stability. Rock mass is quite permeable. Mine water is basic ($pH>7$).
Remnor (Appendix C)	Mine water is acidic ($pH = 3$)	Small pockets of ore for leaching. Low grades (about 0.5% Cu) Loss of gold values (>4.5 g/t Au)

TABLE 1 (Continued)

MINE	ADVANTAGES	DISADVANTAGES
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Copper Mine (Pamour) (Appendix G)	Existing access to a number of small potential test areas (e.g. stopes containing broken ore).	Low copper grades (0.47% Cu) and loss of gold values (1.37 g/t Au). Ground is fractured and heavily foliated in potential leaching zone, and leach solution may migrate. Presence of carbonate (acid-consuming) is a possible disadvantage.

PART III

A copy of the ''Plan of Work and Cost'' follows this page. It illustrates the present time allowance for each Task and the work completed as of December 1, 1984.

Supply and Services Canada
Approvisionnements et Services Canada

CONTRACT PLAN AND REPORT FORM – FORMULE DE PROJET DE CONTRAT ET DE RAPPORT

CONTRACT TITLE – TITRE DU CONTRAT 'PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDE
LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT'

CONTRACTOR – ENTREPRENEUR Centre de Recherche Noranda-NORANDA INC.

TASK DESCRIPTION DESCRIPTION DE LA TÂCHE	1984				TASK DURATION IN MONTHS – DURÉE DE LA TÂCHE EN MOIS				19
	1 July	2 Aug.	3 Sept.	4 Oct.	5 Nov.	6 Dec.	7 Jan.	8 Feb.	9 Ma
1. Mine Site Selection			━━━	━━━	━━━				
2. Pre-feasibility of Underground Operations			┄┄┄	┄┄┄	┄┄━	┄┄┄	┄┄┄		
3. Pre-feasibility of Met. Circuit			┄┄┄	┄┄┄	┄┄━	┄┄┄	┄┄┄		
4. Sensitivity Analysis							┄┄┄	┄┄┄	┄┄┄
PLANNED COST BY MONTH COÛT PRÉVU POUR CHAQUE MOIS			$20,000	$20,000	$15,000	$15,000	$15,000	$10.000	$
ACT. PRÉSEN. E.T.C. BY MONTH PRÉVISION POUR CH. MOIS									

J Martel

S MAS 9143-3 (4/81)

DE CONTRAT ET DE RAPPORT

IBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND
S OF A CANADIAN COPPER SULPHIDE DEPOSIT'
e Recherche Noranda-NORANDA INC.

FILE NO. – Nº DU DOSSIER
07SQ.23110-4-9071

SERIAL NO. – Nº DE SÉRIE

TASK DURATION IN MONTHS – DURÉE DE LA TÂCHE EN MOIS 1985

TASK COST – COÛT DE LA TÂCHE

ct.	5 Nov.	6 Dec.	7 Jan.	8 Feb.	9 Mar.	10 Apr	11 May	12	PLANNED PRÉVU À L'ORIGINE	ACTUAL PRÉSENTEMENT	E.T.C. À VENIR
,000	$15,000	$15,000	$15,000	$10.000	$ 5,000				PLAN		
										ACTUAL COÛTS RÉELS	E.T.C. À VENIR

REFERENCES

Most of the information used to compile this report was gathered from discussions with mine site personnel and from internal reports of the mines involved. Listed below are the published papers which were used:

Ford, R.E. ''Geology of Gaspé Copper Mine'', The Canadian Mining and Metallurgical Bulletin'', July 1959, pp 217-222.

Friesen, R.G., et. al., ''Geology of the Geco Base Metal Deposit'', Precambrian Sulphide Deposits, H.S. Robinson Memorial Volume, edited by R.W. Hutchinson, et. al., Geological Association of Canada.

Geomines Ltd., ''Etude du Drainage de la Fosse du Mont Copper'', Decembre 1975.

Høy, T., et. al., ''Copper-Zinc Deposits Associated with Basic Volcanism, Goldstream Area, Southeastern British Columbia'', Economic Geology, Vol. 79, 1984, pp 789-814.

Humphrey, R.B., ''Mining at Goldstream'', paper presented at the District 6 Meeting of the Canadian Institute of Mining and Metallurgy, Smithers, B.C., October, 1983, 53 p.

MacIsaac, W.F., ''Copper Mountain Geology at Gaspé Copper Mines, Limited'', The Canadian Mining and Metallurgical Bulletin, August 1969.

REFERENCES (Continued)

Sauriol, G., ''Ventilation Requirements in Large Underground Openings at Mines Gaspé'', paper presented at the SME-AIME Fall Meeting and Exhibit, Salt Lake City, Utah, October, 1983.

White, J. C., ''The Development of Mechanized Mining at Gaspé Copper Mines, Limited'', The Canadian Mining and Metallurgical Bulletin, August 1969.

APPENDIX A

MINE SITE REPORT FOR MINES GASPÉ

MACLAREN FOREST PRODUCTS INC.

MINES GASPE

MINE SITE

Location

Mines Gaspé is located in the interior of the Gaspé Peninsula, at the eastern end of the Shick Shock mountain range, and 2 km west of the town of Murdochville, Quebec.

Topography

The local area is mountainous and sharply dissected by deep valleys. Needle Mountain, the highest point locally, reaches an elevation of 915 m. The mine site elevation is about 610 m. Figure 1 shows the topography of the area and Figure 2 is a representative section which also illustrates regional hydrology.

Climatology

Tables 1 and 2 contain average monthly temperature and precipitation values for a 22 year period. The Mines Gaspé area receives an average annual precipitation of 1048.4 mm. Average temperature varies between -13.3°C (January) and 16.3°C (July) in this period, and absolute minimum and maximum values are -29.5°C and 27.9°C respectively.

MINE CONFIGURATION

Orebodies

The underground orebodies underlie the A Zone pit in Needle Mountain (see Figure 3). The lowest of these in the C Zone, which is also the largest, thickest and highest in grade. This ore occurs as a replacement of garnetized limestone which dips as much as 23 degrees to the north, although it is relatively flat at the southern end. The orebody is approximately 915 m long, 34 m thick and varies in width from 152 to 550 m. To the east of the main C Zone, there is a smaller and lower grade orebody in the same limestone horizon. This is referred to as C Zone East. The E-32 deposit is located beneath the town and is presently under development.

Mining Method

Room and pillar mining has been used for most production to date (see Figure 4). Some longhole open stoping has also been done.

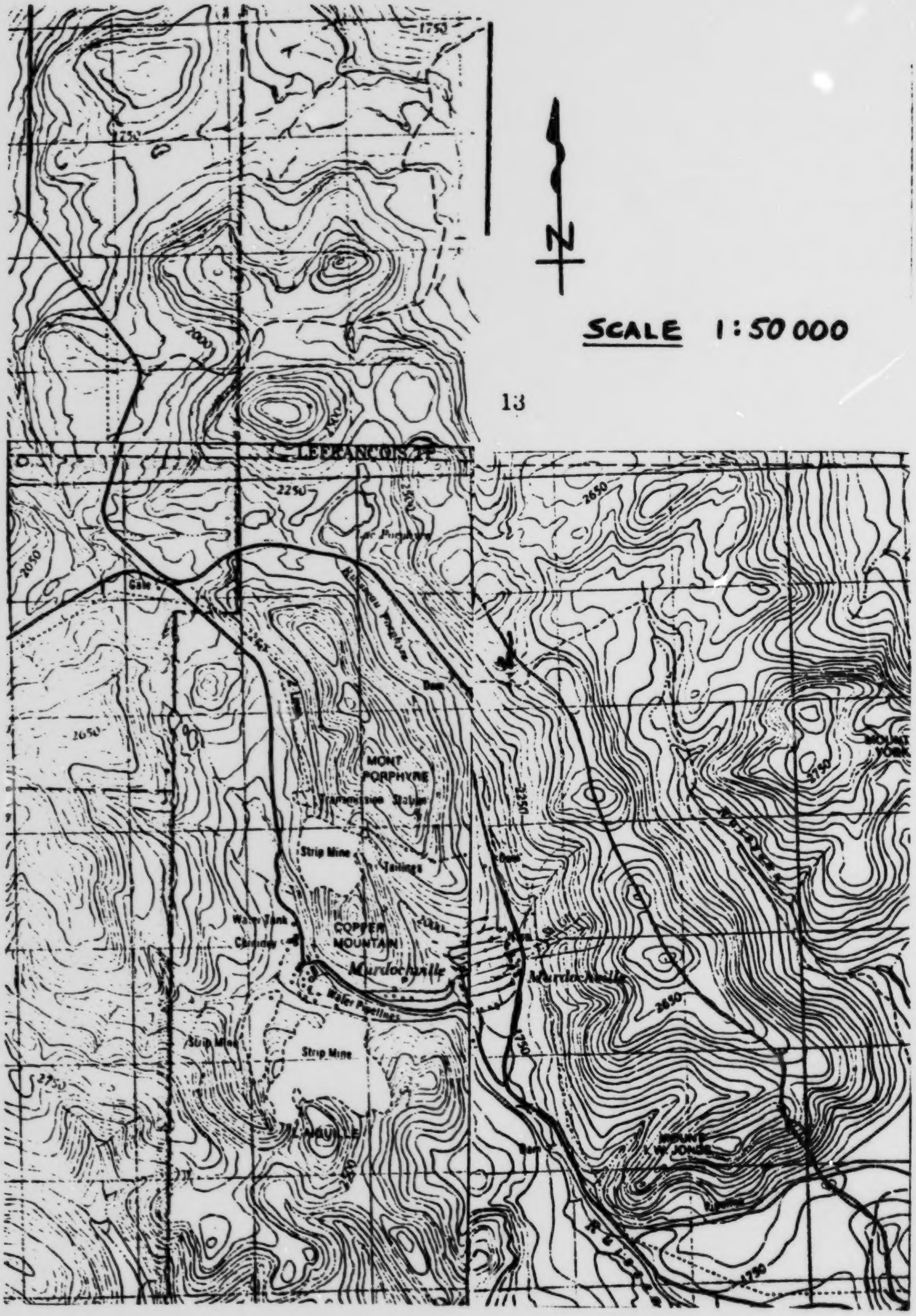
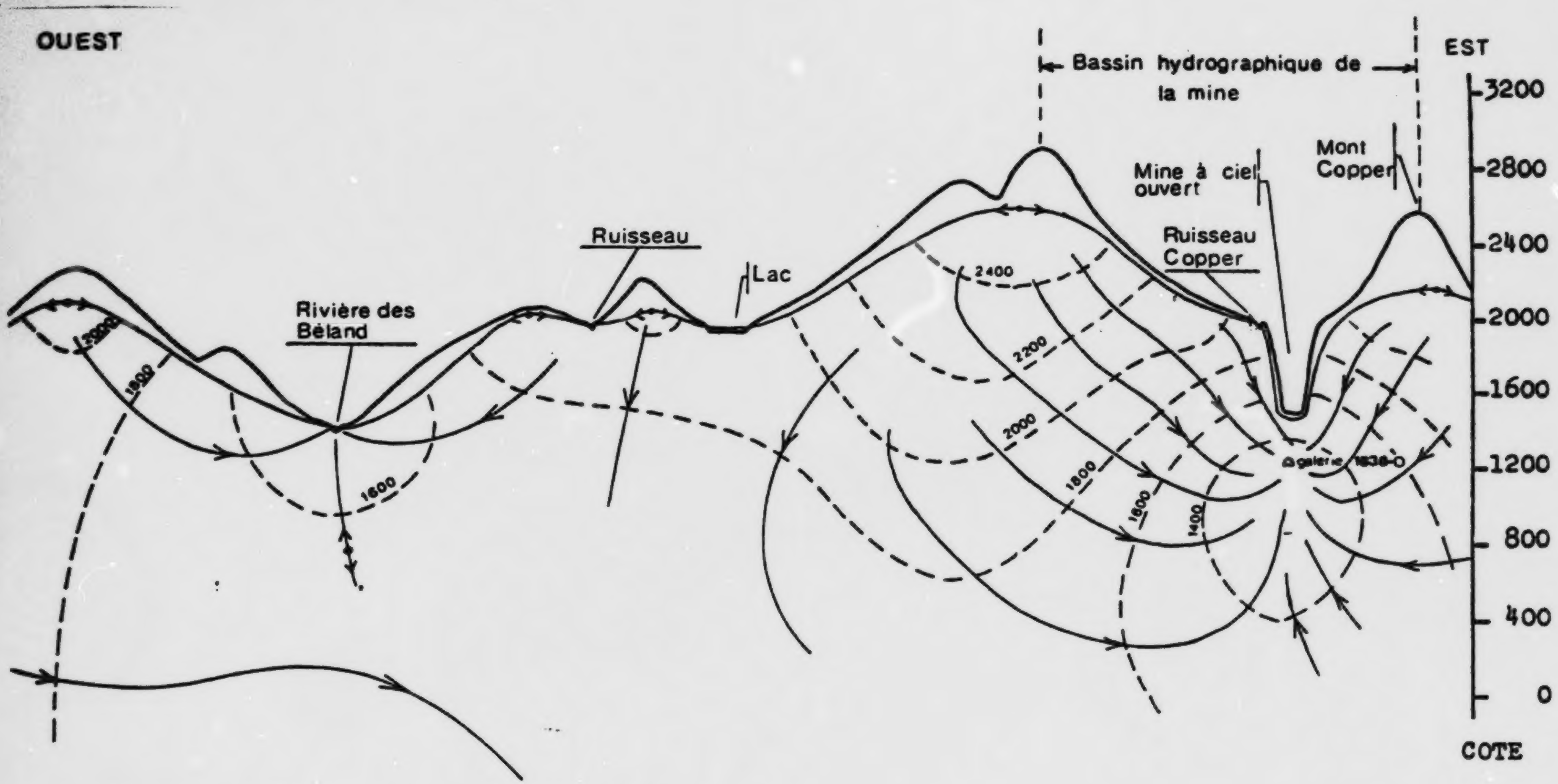

N
SCALE 1:50 000
13
1750
2000
LEFRANCOIS
2250
2650
Gate
Dam
MONT PORPHYRE
Transmission Station
Strip Mine
Tailings
Water Tank
Chimney
COPPER MOUNTAIN
Murdochville
Water Pipelines
Strip Mine
L'AIGUILLE
MOUNT YORK
MOUNT W. W. JONES
2750

Figure 1. Topography of the Area Around Mines Gaspé.


OUEST
EST
Bassin hydrographique de la mine
Mont Copper
Mine à ciel ouvert
Ruisseau Copper
Ruisseau
Lac
Rivière des Béland
galerie 1638-O
2800
2200
1800
1600
2400
2200
2000
1800
1600
1400
3200
2800
2400
2000
1600
1200
800
400
0
COTE

Echelle verticale : 1" = 800' (ou 1:9,600)
Echelle horizontale: 1:50,000
←•→ Point neutre
- - - ligne équipotentielle
——→ ligne de courant

Figure 2. Hydrologic Section of Mines Gaspé Area. After Geomines Ltd., 1975.

TABLE 1

TEMPERATURE DATE - 25 YEAR AVERAGES[1]

MINES GASPE

°C

MONTH	AVERAGE TEMP.	AVERAGE MAXIMUM	AVERAGE MINIMUM	ABSOLUTE MAXIMUM	ABSOLUTE MINIMUM
JANUARY	-13.3	- 8.8	-17.8	2.7	-29.5
FEBRUARY	-12.0	- 7.4	-16.6	2.5	-27.8
MARCH	- 7.2	- 2.8	-11.6	- 6.3	-21.9
APRIL	- 0.8	3.5	- 5.2	14.1	-14.7
MAY	6.5	11.3	1.6	22.4	- 5.4
JUNE	13.1	18.4	7.7	27.1	0.2
JULY	16.3	21.0	11.3	27.9	5.6
AUGUST	16.2	19.3	10.1	27.5	4.1
SEPTEMBER	9.9	14.2	5.6	22.8	- 1.4
OCTOBER	3.6	7.3	0.0	18.1	- 7.4
NOVEMBER	- 2.6	0.9	- 5.8	10.9	-14.9
DECEMBER	- 9.5	- 5.8	-13.3	5.1	-23.9

---

[1] FOR THE PERIOD JANUARY 1959 - DECEMBER 1983

TABLE 2

PRECIPITATION DATA - 25 YEAR AVERAGES[1]

MINES GASPE

MONTH	RAIN mm	SNOW[2] cm	TOTAL PRECIP. mm
JANUARY	8.3	81.1	89.4
FEBRUARY	10.4	56.9	67.3
MARCH	9.2	90.5	99.7
APRIL	21.4	47.6	69.0
MAY	44.9	11.3	56.2
JUNE	83.2	0.0	83.2
JULY	107.9	0.0	107.9
AUGUST	96.5	0.0	96.5
SEPTEMBER	75.7	0.6	76.3
OCTOBER	69.2	17.6	86.8
NOVEMBER	37.0	55.8	92.8
DECEMBER	20.9	102.4	123.3
TOTAL	584.6	463.8	1048.4

[1] FOR THE PERIOD JANUARY 1959 - DECEMBER 1983

[2] 1 cm SNOW ≡ 1 mm RAIN

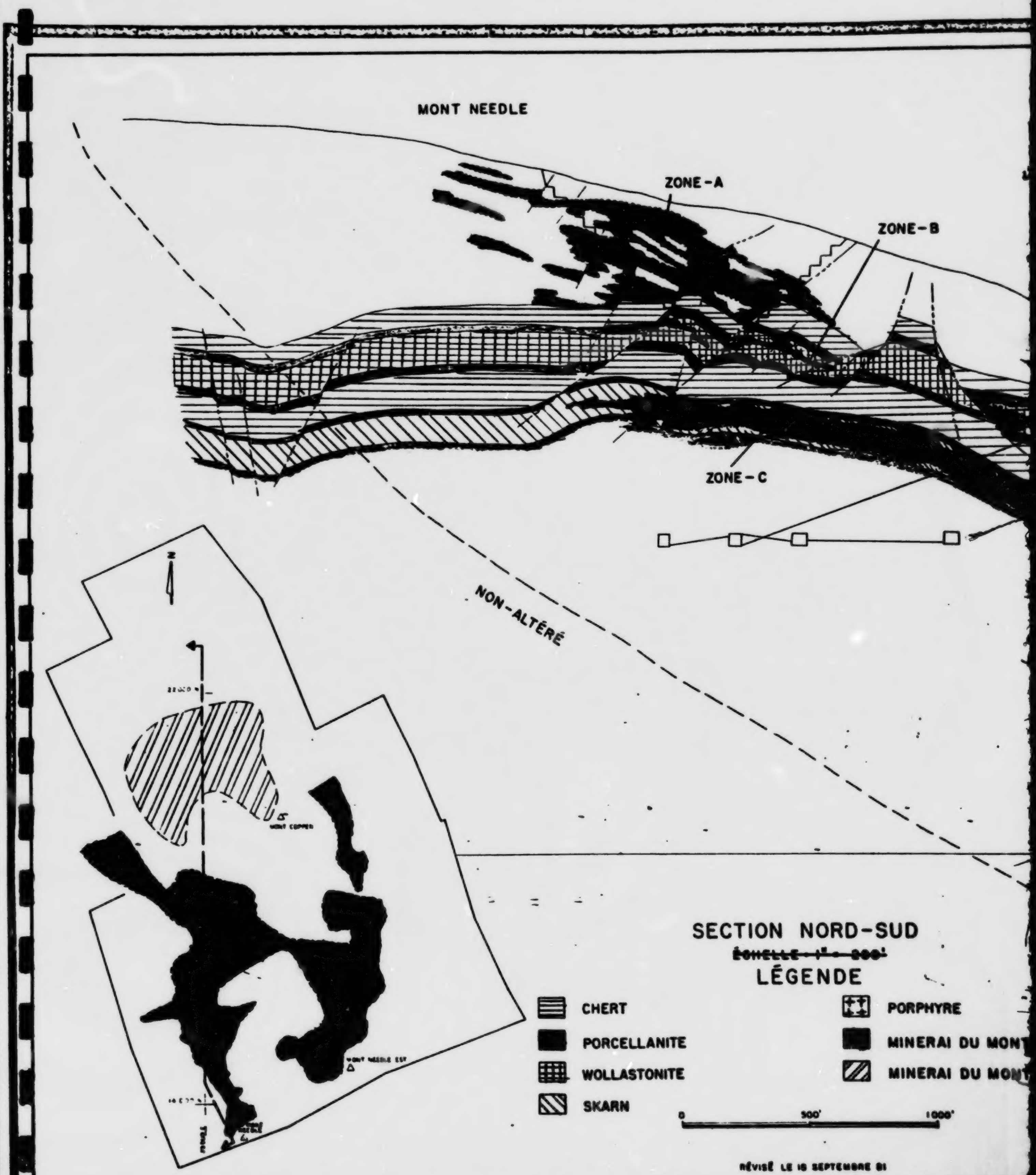
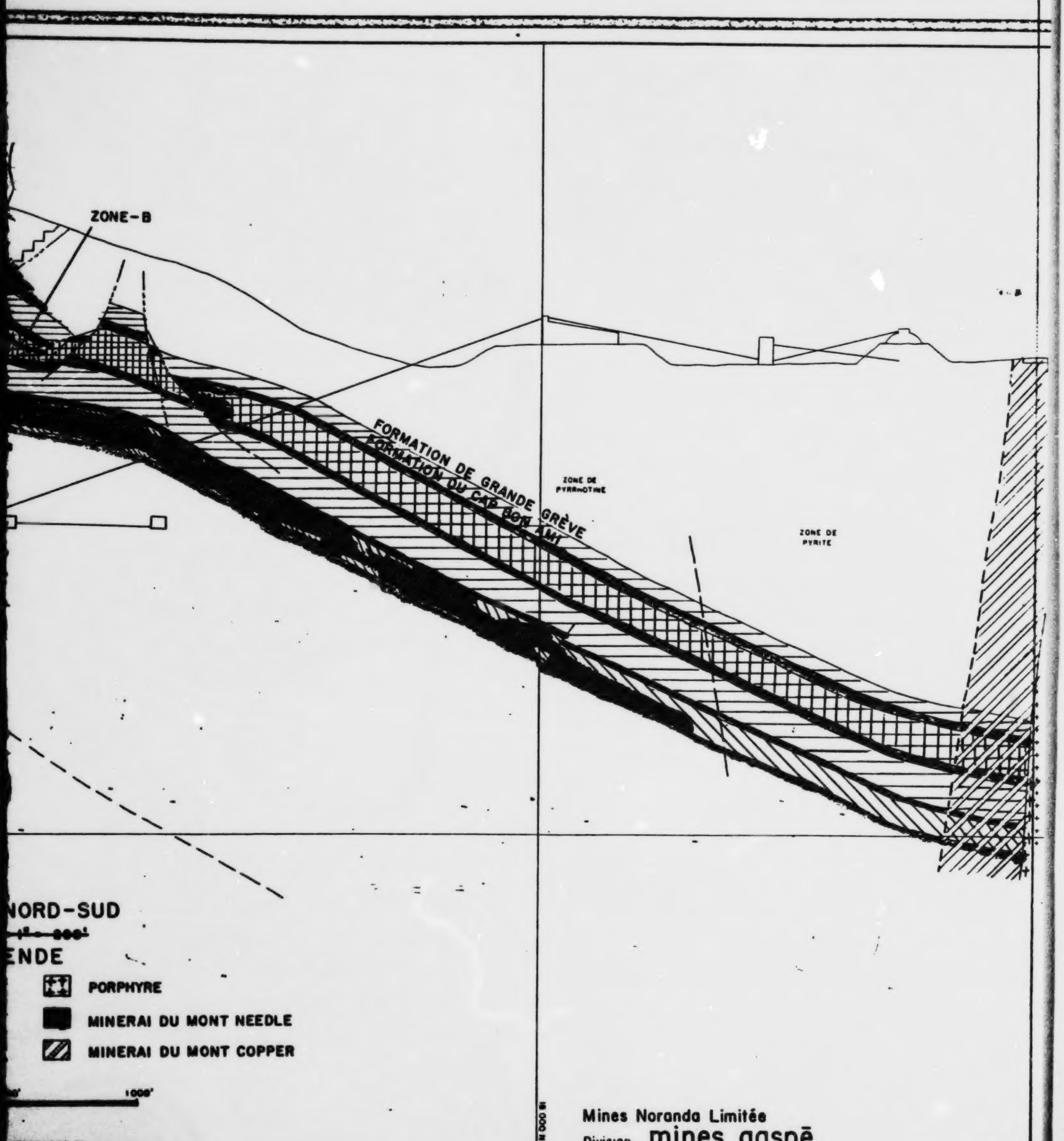

MONT NEEDLE
ZONE-A
ZONE-B
ZONE-C
NON-ALTÉRÉ
N
MONT COPPER
MONT NEEDLE EST
SECTION NORD-SUD
~~ÉCHELLE 1" = 200'~~
LÉGENDE
CHERT
PORCELLANITE
WOLLASTONITE
SKARN
PORPHYRE
MINERAI DU MONT
MINERAI DU MONT
0
500'
1000'
RÉVISÉ LE 10 SEPTEMBRE 81

Figure 3. Geological Section of Mines


ZONE-B
FORMATION DE GRANDE GRÈVE
FORMATION DU CAP BON AMI
ZONE DE PYRRHOTINE
ZONE DE PYRITE
NORD-SUD
ENDE
PORPHYRE
MINERAI DU MONT NEEDLE
MINERAI DU MONT COPPER
1000'
SEPTEMBRE 84
Mines Noranda Limitée
Division mines gaspē

ical Section of Mines Gaspé Orebodies.

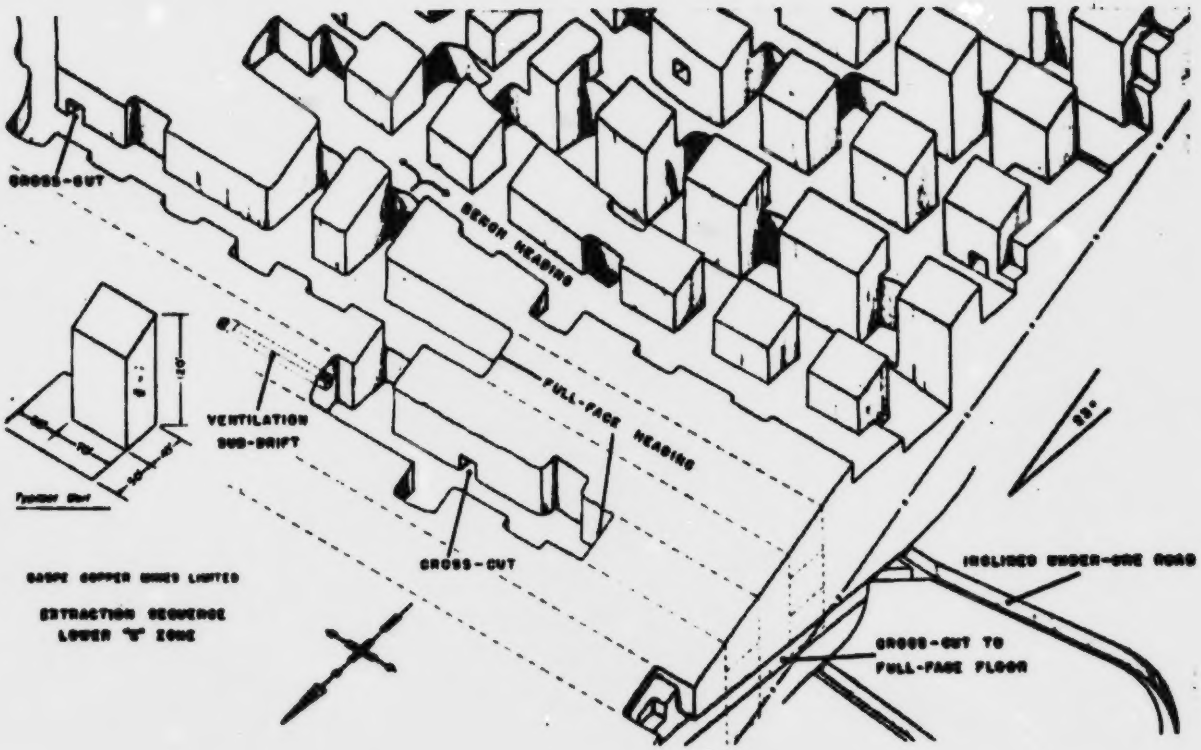

CROSS-CUT
BENCH HEADING
FULL-FACE HEADING
VENTILATION SUB-DRIFT
CROSS-CUT
INCLINED UNDER-ORE ROAD
CROSS-CUT TO FULL-FACE FLOOR
Typical Unit
GASPE COPPER MINES LIMITED
EXTRACTION SEQUENCE
LOWER "C" ZONE

Figure 4. Room and Pillar Mining in the Lower C Zone, Mines Gaspé. After White, 1968.

Tonnage

Milling capacity is 30,700 tonnes per day for sulphide ore. Vat leaching capacity is 4,500 tonnes per day.

Surface Facilities

Surface facilities at Mines Gaspé consist of two concentrators, a smelter, a vat leaching plant for oxide copper, a precipitation plant, an acid plant, shops and a lab. Figure 5 illustrates the surface layout at Mines Gaspé.

GEOLOGY

Mineralogy

An east-west section through the Needle Mountain orebodies is shown in Figure 3. Figure 6 shows the location of the orebodies in the stratigraphic column.

Orebodies:	Chalcopyrite is the predominant copper mineral. Pyrrhotite occurs in nearly equal proportions. Molybdenite and scheelite occur throughout the orebodies, and bismuth occurs in solid solutions with chalcopyrite. Silver, gold and selenium are also present. Bornite, chalcocite and tennantite occur as minor minerals in the C Zone.
Skarn:	Major minerals are pyroxene, grossularite and wollastonite. Minor minerals are chlorite, muscovite, epidote, clinozoisite, allanite and thompsonite.
Porcellanite:	diopsidic porcellanite
Wollastonite:	wollastonite diopside

Chemical Analysis

Table 3 contains chemical analyses for ore from the C North-West Zone and the Murdochville project.

Structure

Folding:	The general structure is that of closed folding. The anticlinal axis is on the crest of Needle Mountain and the synclinal axis is half a mine north of Copper Mountain.

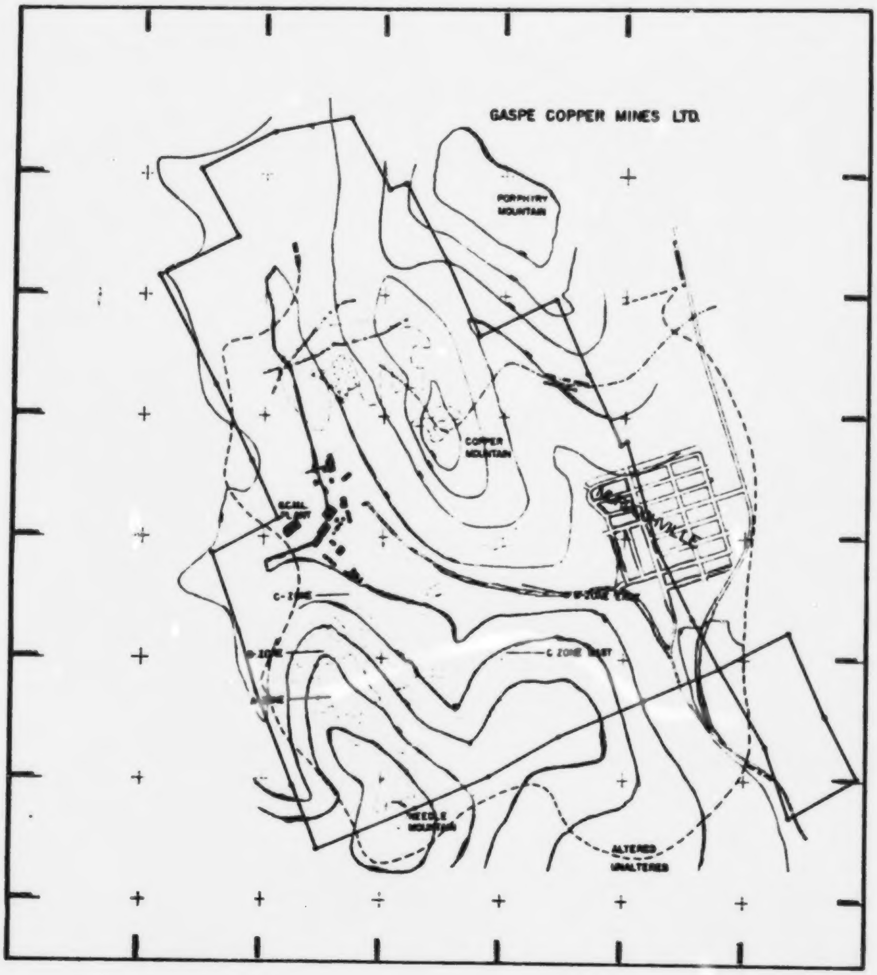
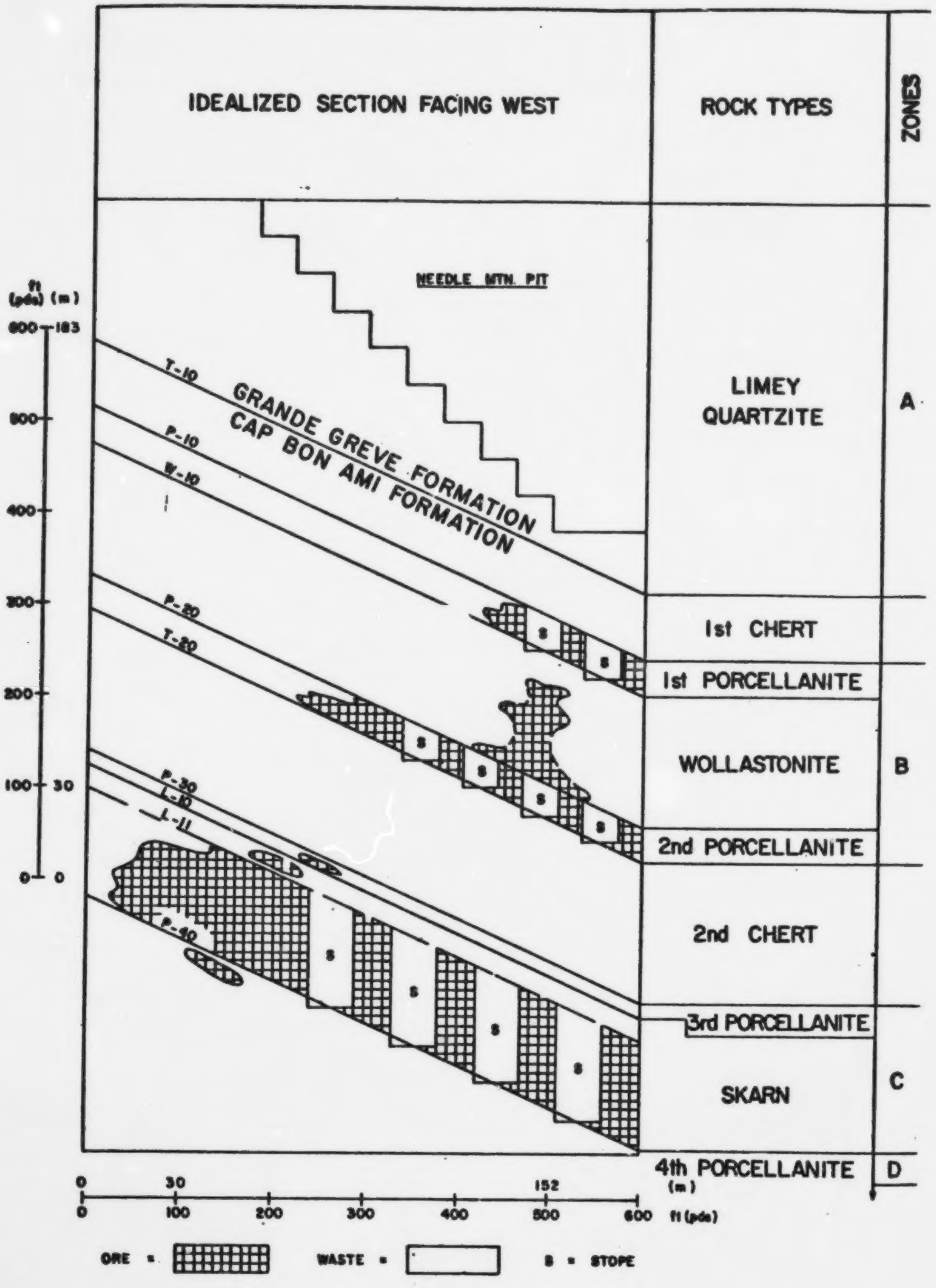

GASPE COPPER MINES LTD.
PORPHYRY MOUNTAIN
COPPER MOUNTAIN
C-ZONE
C ZONE EAST
NEEDLE MOUNTAIN
ALTERED
UNALTERED

Figure 5. Surface Layout at Mines Gaspé, After MacIsaac, 1969.



IDEALIZED SECTION FACING WEST
ROCK TYPES
ZONES
NEEDLE MTN. PIT
ft (pds) (m)
600
183
500
400
300
200
100
30
0
0
T-10
GRANDE GREVE FORMATION
CAP BON AMI FORMATION
P-10
W-10
P-20
T-20
P-30
L-10
L-11
P-40
S
LIMEY QUARTZITE
A
1st CHERT
1st PORCELLANITE
WOLLASTONITE
B
2nd PORCELLANITE
2nd CHERT
3rd PORCELLANITE
SKARN
C
4th PORCELLANITE
D
0
30
152
(m)
0
100
200
300
400
500
600
ft (pds)
ORE =
WASTE =
S = STOPE

Figure 6. Sequence of Rock Units and Ore Zones - Needle Mountain.

TABLE 3
CHEMICAL ANALYSIS OF MINES GASPE ORE

		NEEDLE MOUNTAIN ORE (C NORTH-WEST ZONE)	MURDOCHVILLE PROJECT ORE
Cu	%	1.63	1.82
Mo	%	0.12	0.006
Au	g/t	0.05	0.05
Ag	g/t	5.14	11.66
Pb	%	0.005	0.009
Zn	%	0.019	0.11
Bi	%	0.007	0.014
Fe	%	10.69	8.00
Mn	%	0.055	0.049
Ni	%	0.023	0.04
Cr	%	0.019	0.05
Cd	%	0.0004	0.002
Sb	%	0.03	0.01
As	%	0.011	0.008
Se	%	0.028	0.01
W	%	0.014	0.01
Te	%	0.0194	0.006
Hg	ppm	0.12	0.06
S	%	2.08	3.80
SiO_2	%	40.92	48.10
CaO	%	23.50	11.19
MgO	%	3.20	4.84
Al_2O_3	%	6.80	5.37

Faulting:	Four major normal fault systems have been encountered underground. Three of these systems dip at 60° to the south, and the most northerly fault dips steeply north. All of the faults horse-tail and die rapidly, both at depth and laterally.
Jointing:	Regional jointing is very prominent. The predominant joints strike $N10^\circ W$ (at right angles to the trend of the general structure) and dip steeply to vertically.
RQD:	RQD values exceed 90 percent.

MINE WATER

Figure 7 is a schematic of the underground pumping network at Mines Gaspé showing pump capacities. The average volume of water pumped per month is shown in Figure 8. Peak volumes are recorded in the spring months, corresponding to peak rainfall figures (see Table 2).

Studies conducted by Geomines Ltée. in 1975 and 1978 indicated a strong correlation between precipitation and volume of water pumped from the mine, and reported "moderate" permeability of 8.7×10^{-5} cm/sec for the porphyry material beneath the Copper Mountain pit. The permeability was provided by a multitude of fractures in the rock mass. These data indicate the potential of environmental contamination by leach solutions.

Table 4 contains an example of a mine water analysis. Negligible mercury is present. A major factor, however, is the relatively basic nature of the water ($pH > 7$). The mine pumping network is not equipped to handle the acidic water conditions that would be caused by leaching.

VENTILATION

335 m^3/s of fresh air is supplied to the mine workings. In winter, the fresh air is heated to $2^\circ C$. Air temperature underground is about $8^\circ C$ at 95% relative humidity.

POTENTIAL ZONES FOR BACTERIAL LEACHING

Ore pillars in the B and C Zones are the major potential source of material for leaching tests. Figures 9 and 10 illustrate the size, shape and placement of these pillars. Approximately 10.4 million tonnes of material grading 1.57% Cu is contained in these pillars.

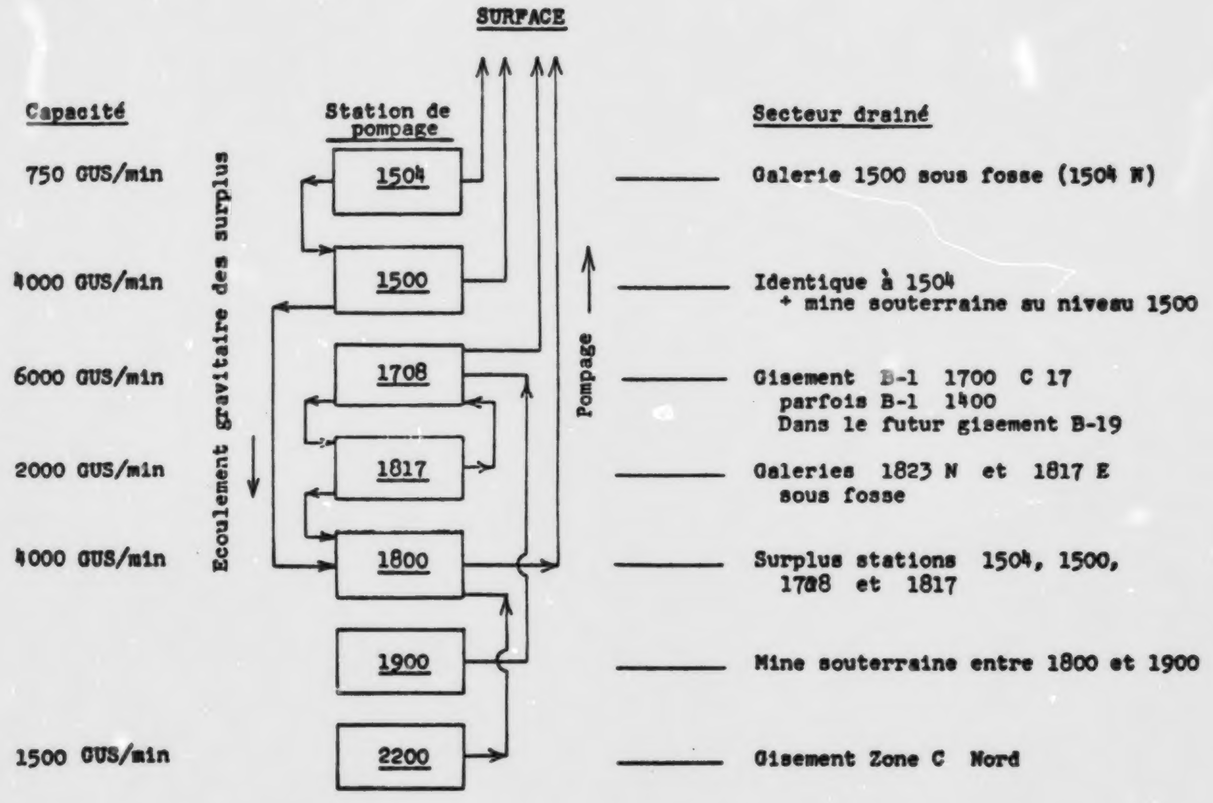
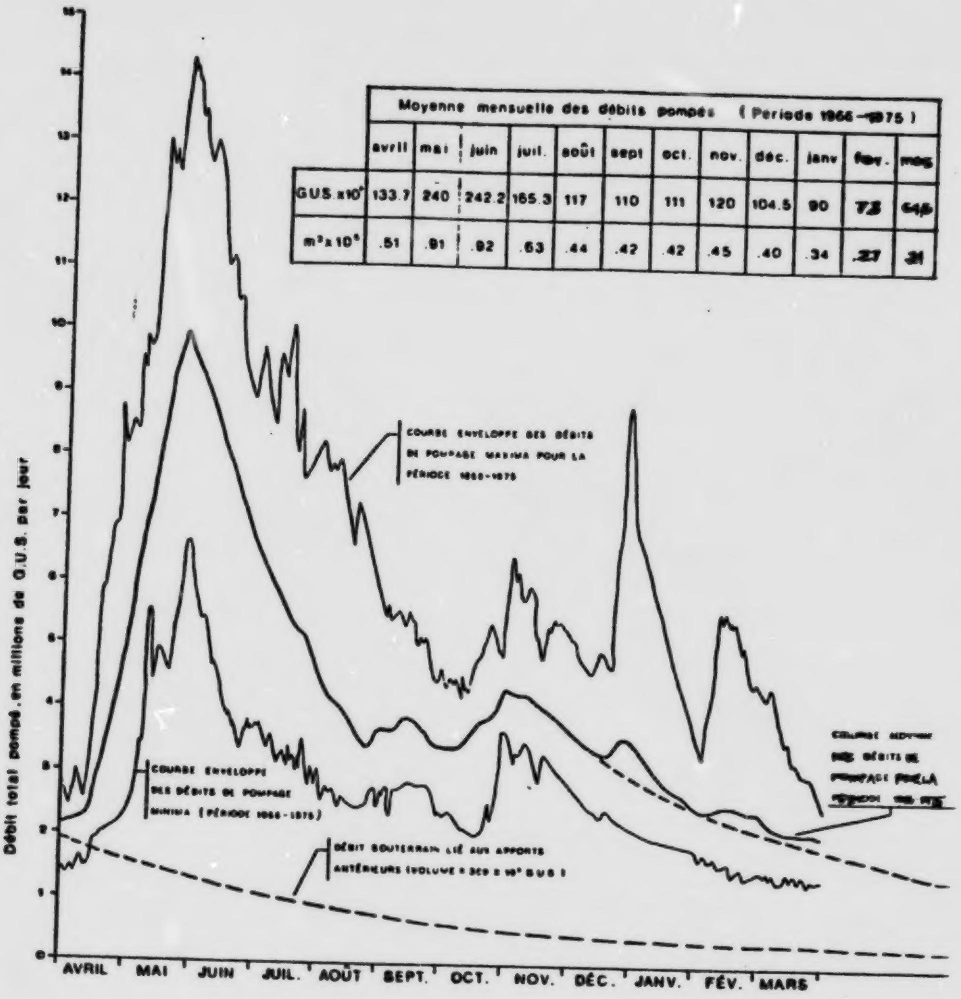

SURFACE
Capacité
Station de pompage
Secteur drainé
750 GUS/min
1504
Galerie 1500 sous fosse (1504 N)
4000 GUS/min
1500
Identique à 1504
+ mine souterraine au niveau 1500
6000 GUS/min
1708
Gisement B-1 1700 C 17
parfois B-1 1400
Dans le futur gisement B-19
2000 GUS/min
1817
Galeries 1823 N et 1817 E
sous fosse
4000 GUS/min
1800
Surplus stations 1504, 1500,
1708 et 1817
1900
Mine souterraine entre 1800 et 1900
1500 GUS/min
2200
Gisement Zone C Nord
Ecoulement gravitaire des surplus
Pompage

Figure 7. Schematic of Underground Pumping Installations, After Geomines Ltd., 1975.



	Moyenne mensuelle des débits pompés (Periode 1966–1975)											
	avril	mai	juin	juil.	août	sept	oct.	nov.	déc.	janv	fev.	mars
GUS.x10^6	133.7	240	242.2	165.3	117	110	111	120	104.5	90	73	64.5
m^{3}x10^6	.51	.91	.92	.63	.44	.42	.42	.45	.40	.34	.27	.21

Figure 8. Volume Pumped to Surface Per Day, 1966-75, After Geomines, Ltd., 1975.

TABLE 4

ANALYSIS OF MINE WATER-MONTHLY AVERAGES
MINES GASPE
(mg/1)

MONTH	pH	Cu-T[1]	Cu-D[2]	Fe-T	Fe-D	Mn-T	Mn-D	SOLS	HARD
APRIL 1983	7.60	0.36	0.08	0.80	0.05	0.125	0.075	30.0	333
MAY 1983	7.85	0.09	0.03	0.20	0.02	0.075	0.050	6.0	368
AUGUST 1983	7.95	0.08	0.01	0.25	0.02	0.075	0.075	1.0	425
SEPTEMBER 1983	7.85	0.06	0.03	0.10	0.02	0.075	0.050	0.5	453

[1] T = TOTAL

[2] D = DISSOLVED

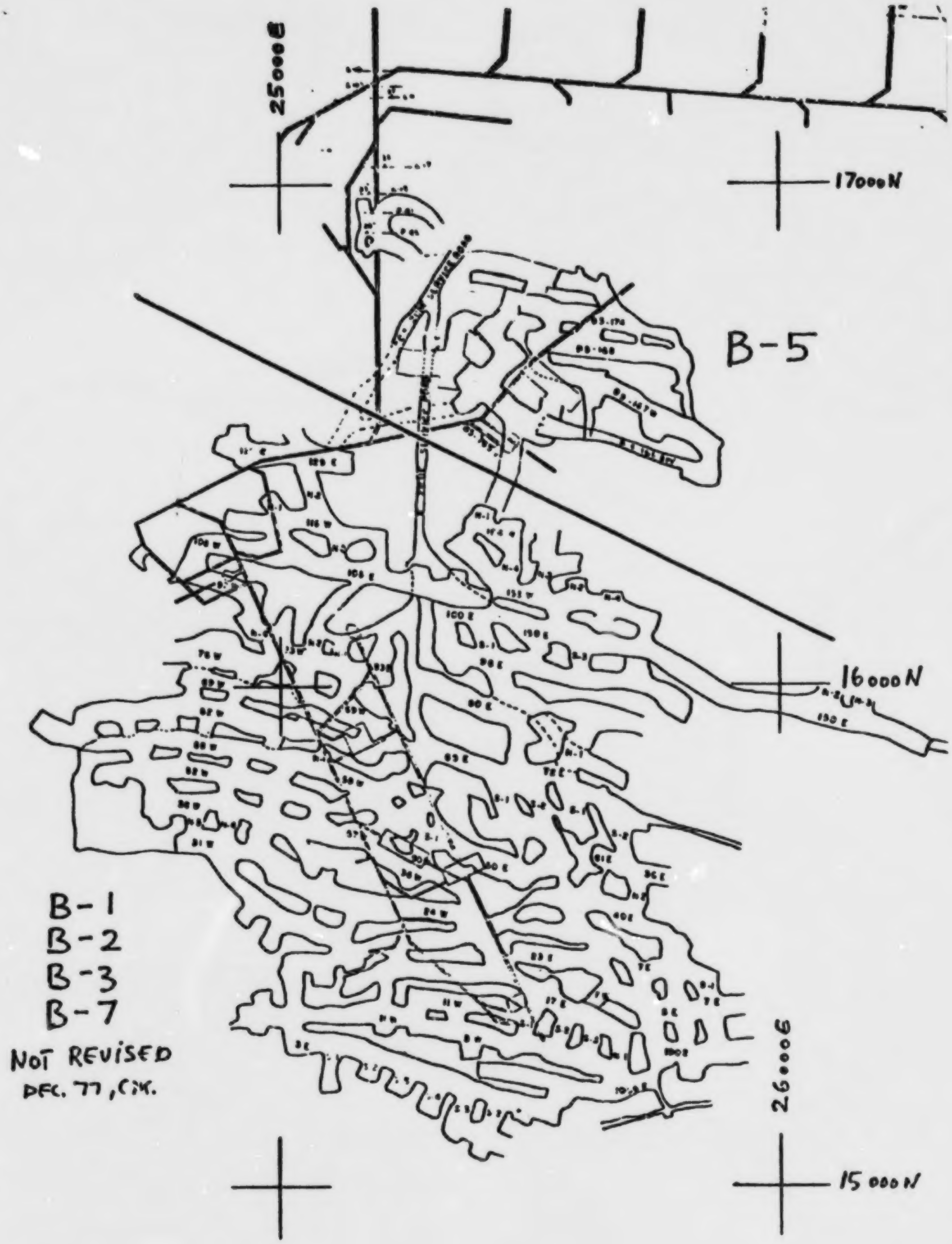
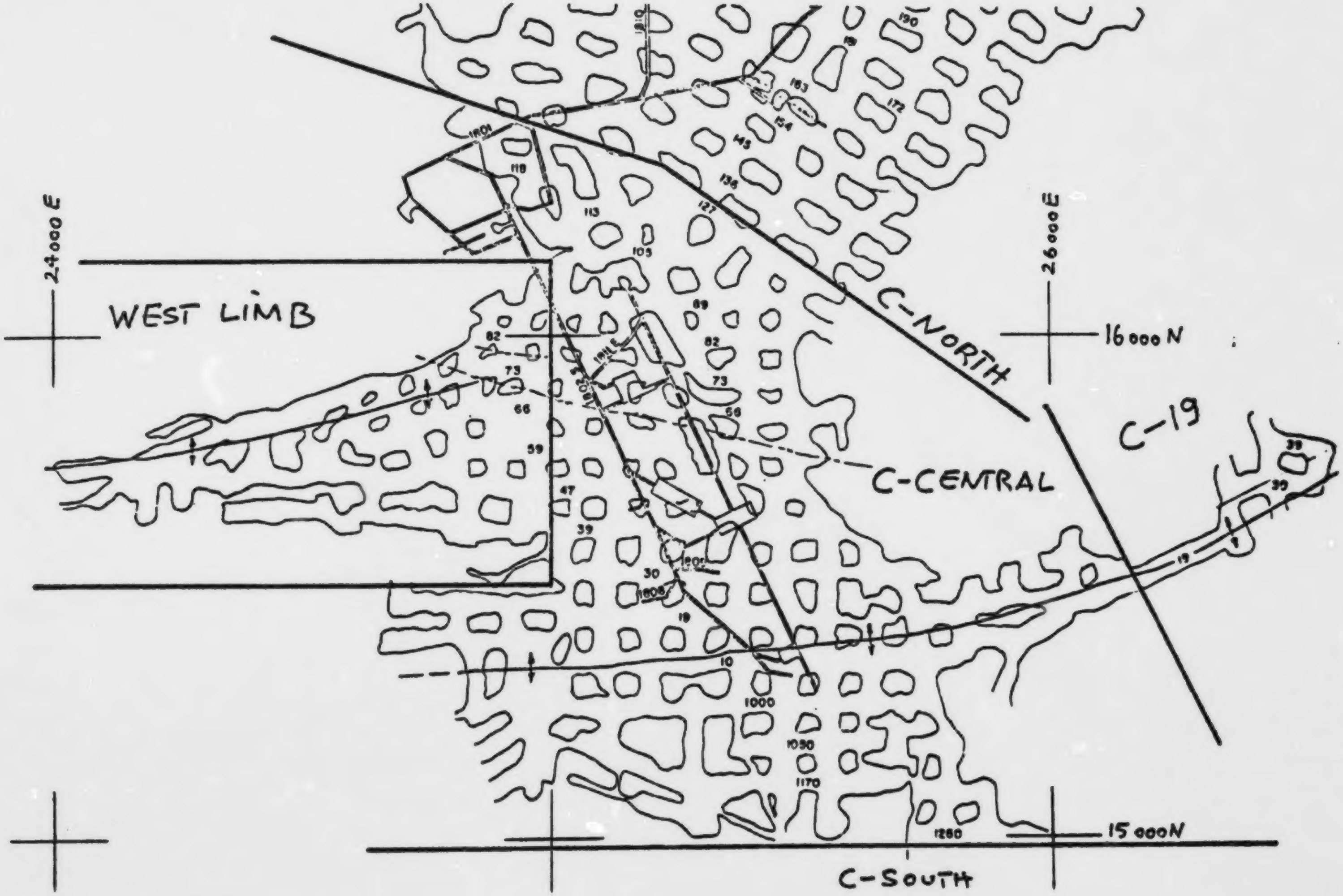

25000E
17000N
B-5
16000N
26000E
15000N
B-1
B-2
B-3
B-7
NOT REVISED
DEC. 77, CJK.

Figure 9. Example of Pillar Layout in the B Zone.



24000E
WEST LIMB
26000E
16000N
C-NORTH
C-19
C-CENTRAL
C-SOUTH
15000N
190
172
163
154
145
136
127
118
113
105
89
82
73
66
59
47
39
30
19
10
1000
1030
1170
1280
39

Figure 10. Example of Pillar Layout in the C Zone.

APPENDIX B

MINE SITE REPORT FOR THE HEATH STEELE MINE

HEATH STEELE MINES LIMITED

MINE SITE

Location

Heath Steele Mines Limited is located in Northern New Brunswick, 52 km north-west of Newcastle.

Topography

The area around the mine site has relatively gentle topography. The highest point locally is a hill to the north, at 387 m elevation. The mine site is at about 350 m elevation. Figure 1 contains north-south and east-west sections through the mine site.

Climatology

Average temperature data from 1982 and 1983 are presented in Table 1. Average temperatures for these two years varied between - 12°C (January) and 18.5°C (July). Absolute minimum and maximum values for this period are -29.5°C and 33.3°C. Average annual precipitation (see Table 2) is 1145.5 mm.

MINE CONFIGURATION

Orebodies

The Heath Steele orebodies occur in an east-west trending belt between a footwall of chloritic tuff and a hanging wall of quartz feldspar porphyry. Dips are steeply north to vertical, the orebodies vary in strike length from 1120 to 1830 m and average 18 m in width (but may exceed 46 m in width). From east to west the Heath Steele orebodies are identified as B, E, A-C-D and West Grid.

Mining Method

The mining method is large diameter longhole open stoping. Partial filling is carried out using development waste and rock from a surface pit. Some test work has been done using mill tailings for backfill. Stopes are 55 to 76 m-high. Figure 2 shows a north-south long section through the mine.

Tonnage

Mill capacity is 3900 tonnes per day. A hoisting rate of 5440 tonnes per day (ore) is used when the mine is in production. Heath Steele is currently shut down.

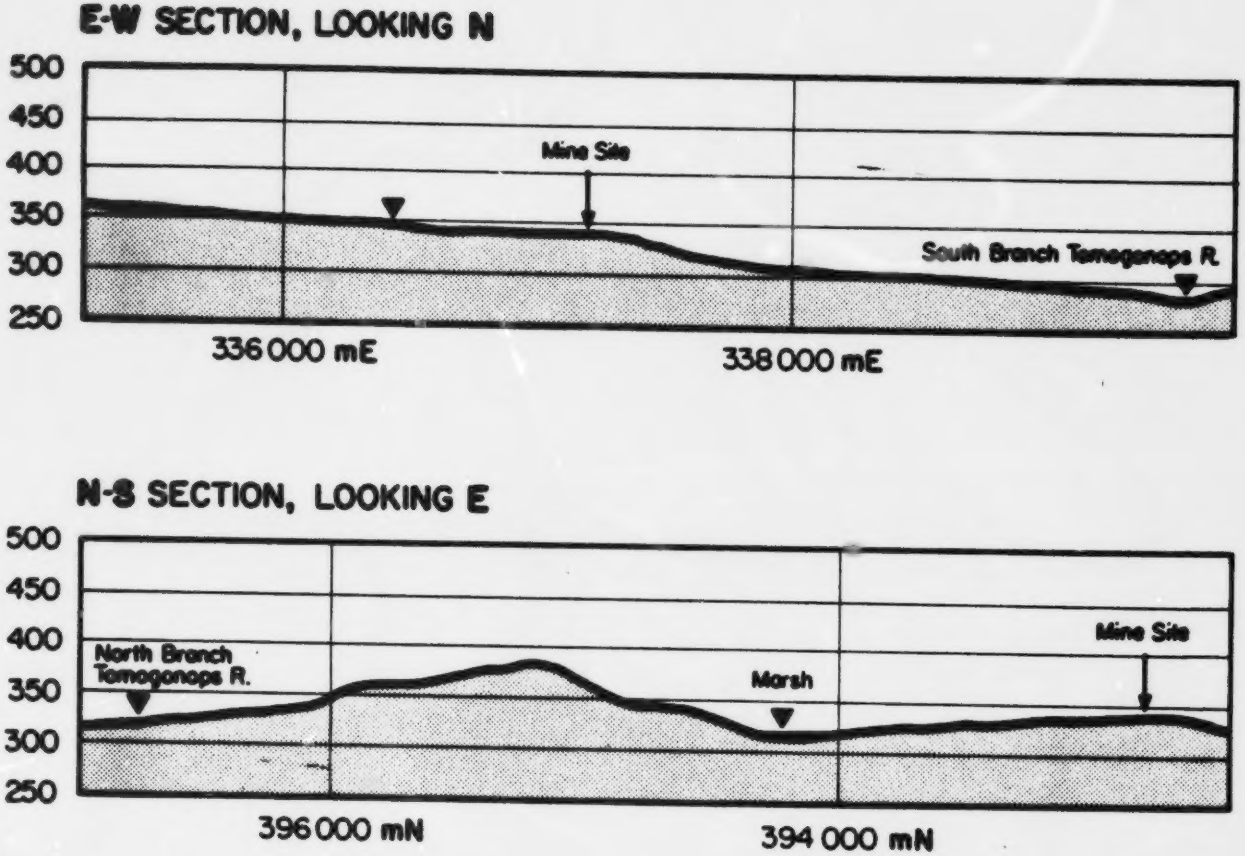


Figure 1. Topographic Sections Through the Heath Steele Mine Site.

TABLE 1

TEMPERATURE DATA - AVERAGE OF 2 YEARS[1]
HEATH STEELE
(°C)

MONTH	AVERAGE TEMP.	AVERAGE MAXIMUM	AVERAGE MINIMUM	ABSOLUTE MAXIMUM	ABSOLUTE MINIMUM
JANUARY	-12.0	- 7.3	-16.6	5.8	-29.5
FEBRUARY	-10.7	- 6.3	-15.1	2.8	-23.5
MARCH	- 4.8	- 0.5	- 9.1	9.0	-20.3
APRIL	1.4	6.2	- 3.5	19.8	-19.5
MAY	8.7	13.9	3.4	26.0	- 1.5
JUNE	14.9	20.9	8.9	29.8	1.5
JULY	18.5	23.9	13.0	33.3	7.0
AUGUST	16.2	21.1	11.4	28.0	5.0
SEPTEMBER	13.2	18.1	8.3	29.5	- 0.3
OCTOBER	6.4	10.9	1.8	21.3	- 8.3
NOVEMBER	0.2	3.2	- 3.7	16.5	-14.0
DECEMBER	- 7.5	- 2.9	-12.0	8.8	-25.0

[1] 1982 AND 1983

TABLE 2

PRECIPITATION DATA - 24 YEAR AVERAGES[1]
HEATH STEELE

MONTH	RAIN mm	SNOW[2] cm	TOTAL PRECIP. mm
JANUARY	14.0	72.7	86.7
FEBRUARY	9.7	58.9	68.6
MARCH	23.5	68.3	91.8
APRIL	45.0	41.1	86.1
MAY	90.1	5.2	95.3
JUNE	89.5	0.0	89.5
JULY	98.8	0.0	98.8
AUGUST	95.1	0.0	95.1
SEPTEMBER	98.5	0.2	98.7
OCTOBER	103.1	3.9	107.0
NOVEMBER	72.5	35.4	107.9
DECEMBER	37.7	82.3	120.0
TOTAL	777.5	368.0	1145.5

[1] 1960 - 1983

[2] 1 cm SNOW ≡ 1 mm RAIN

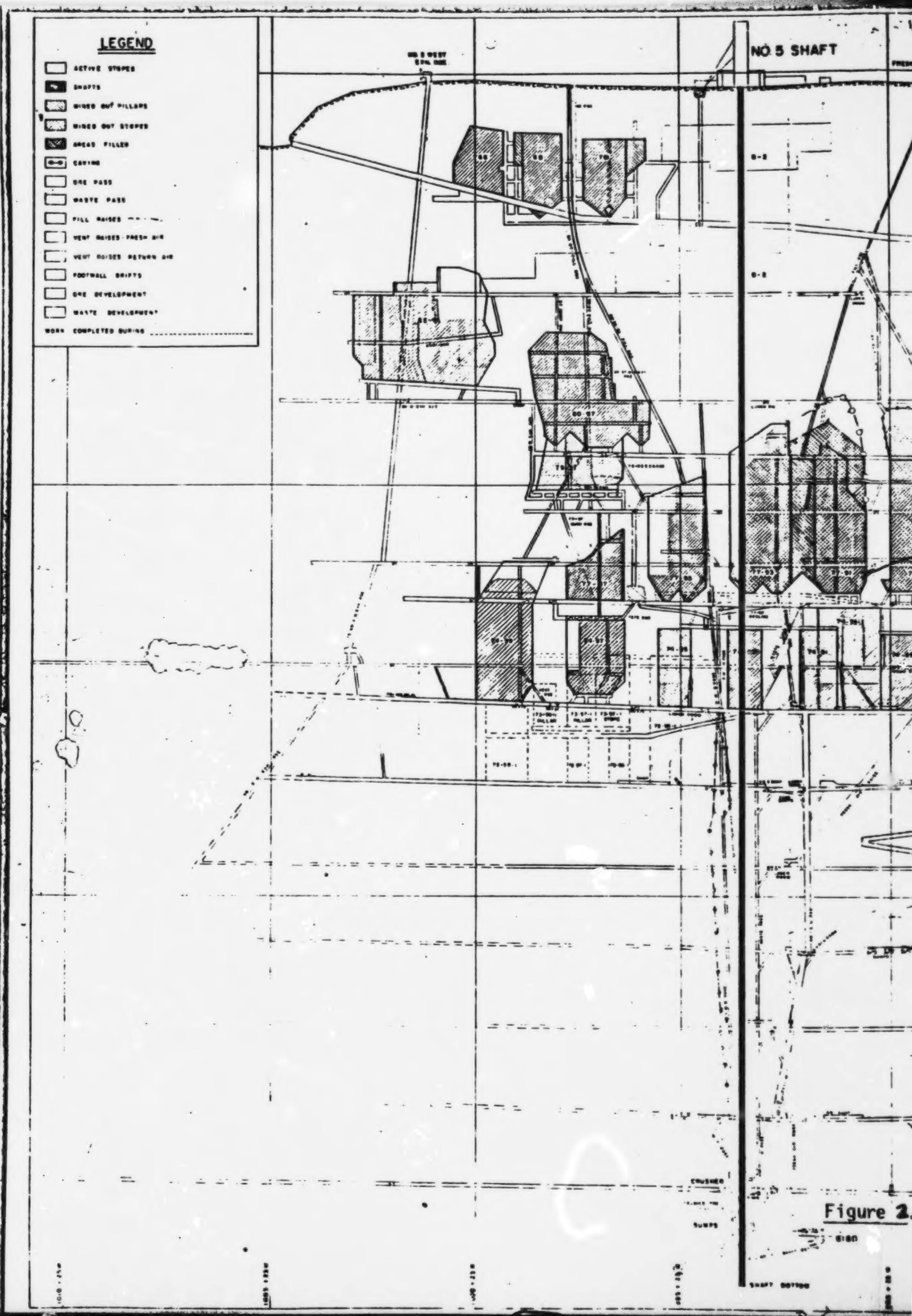

LEGEND
ACTIVE STOPES
SHAFTS
MINED OUT PILLARS
MINED OUT STOPES
AREAS FILLED
CAVING
ORE PASS
WASTE PASS
FILL RAISES
VENT RAISES FRESH AIR
VENT RAISES RETURN AIR
FOOTWALL DRIFTS
ORE DEVELOPMENT
WASTE DEVELOPMENT
WORK COMPLETED DURING
NO.5 SHAFT
CRUSHER
SUMPS
SHAFT BOTTOM

Figure 2.

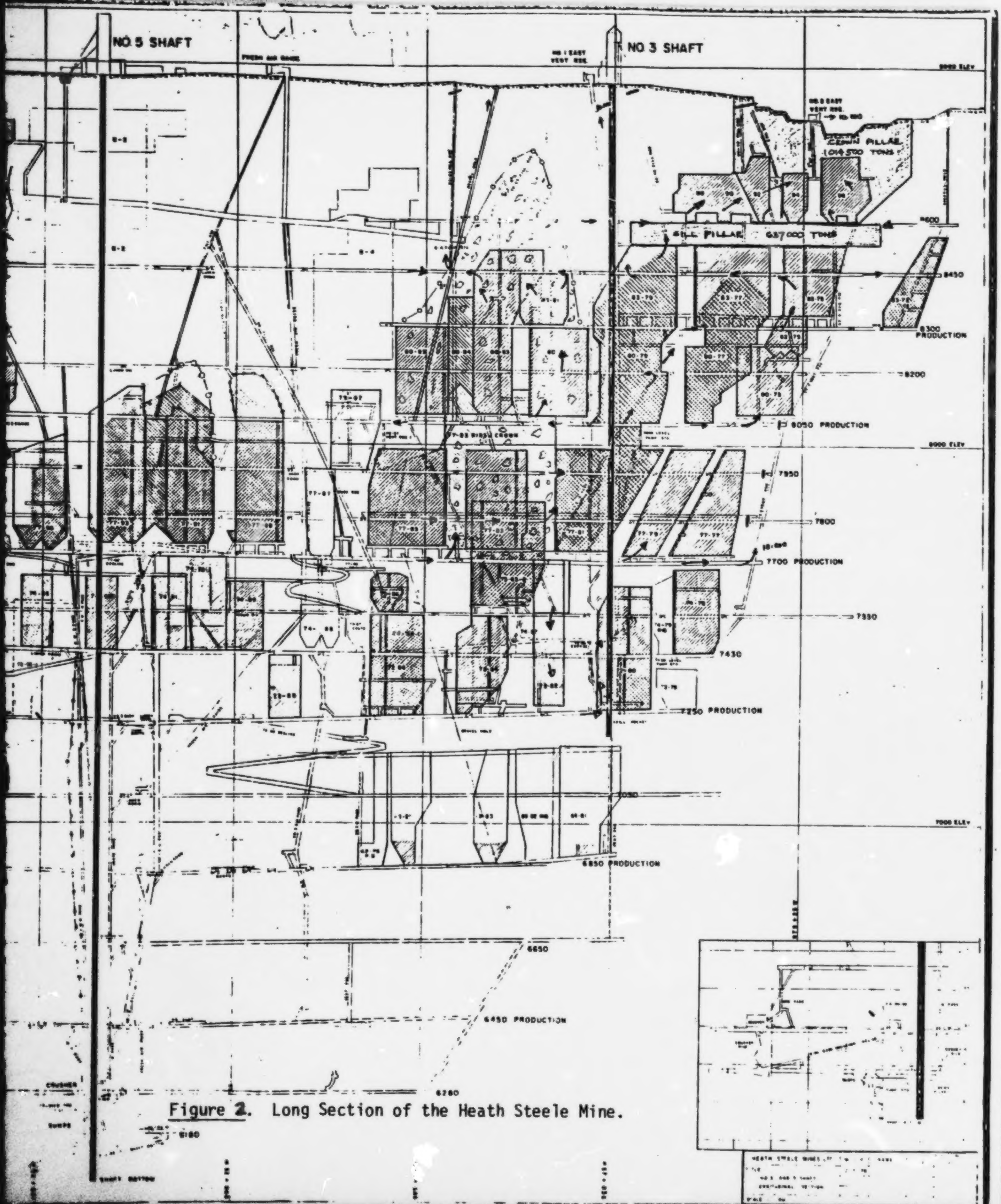

NO.5 SHAFT
NO 3 SHAFT
9000 ELEV
CROWN PILLAR
1,014,500 TONS
SILL PILLAR 637,000 TONS
8600
8450
8300 PRODUCTION
8200
8050 PRODUCTION
8000 ELEV
7950
7800
7700 PRODUCTION
7550
7430
7250 PRODUCTION
7050
7000 ELEV
6850 PRODUCTION
6650
6450 PRODUCTION
6280
6180
CRUSHER
DUMPS
SHAFT BOTTOM
HEATH STEELE MINES LTD
LONGITUDINAL SECTION

Figure 2. Long Section of the Heath Steele Mine.

Surface Facilities

Surface facilities consist of a mill which produces copper, lead and zinc concentrates by flotation.

GEOLOGY

Mineralogy

The ore zones comprise 95% massive sulphides and 5% gangue. In the massive sulphides, the primary ore minerals are sphalerite, galena, chalcopyrite, magnetite, pyrite and pyrrhotite with minor marcasite, bismuthite, hematite, tetrahedrite-tennantite and pyrargyrite. Secondary ore minerals are chalcocite, covellite and marcasite with minor native silver and pyrargyrite. The gangue minerals are quartz, chlorite, kaolinite and carbonates. The hangingwall contains quartz, feldspar porphyry and quartz porphyry, and the footwall contains argillite, graywacke and chert.

Base metal distribution in the B Zone indicates that the copper and lead-zinc ores occupy characteristic positions within the massive sulphides, creating mineralogical zoning across and along the strike of the deposit. Normally, the copper-rich ore lies conformably on the stratigraphic bottom of the deposit and the lead-zinc ore lies at the top. In addition, the east portion of the B Zone is rich in lead-zinc and the west in copper.

Chemical Analysis

Table 3 contains a chemical analysis of Heath Steele mill feed.

Structure

There have been five major periods of folding in the area which have produced a structurally complex orebody. The B Zone is folded into a series of very tight isoclinal folds which plunge at about 60° in a S73°W direction. These sets of folds have well developed fracture systems.

Heath Steele has experienced problems with pillar failure due to high local stresses, and with caving above mined-out stopes.

Ambient ground temperature is 5 - 8°C.

TABLE 3

CHEMICAL ANALYSIS OF HEATH STEELE MILL FEED

Cu	%	0.85
Pb	%	1.36
Zn	%	4.18
Fe	%	33.70
S	%	34.70
As	%	0.19
Bi	%	0.041
Sb	%	0.016
Cd	%	0.009
Sn	%	0.035
Ni	%	0.002
Co	%	0.070
Insol.	%	14.44
Ag	g/t	51.20
Au	g/t	1.09

MINE WATER

Table 4 contains a chemical analysis of mine water pumped from the B-4 pump station which includes water draining from the area below the B Zone pit. Averaged 1981 and 1982 pumping volumes by month are contained in Table 5. Data on flow paths and rock mass permeability are not available.

The mine water is naturally acidic (pH< 4.0) and the pumping network is equipped to handle these conditions. In-situ leaching would not cause any additional problems from this point of view.

Mine water at Heath Steele is treated and monitored to prevent any contamination of the nearby Miramichi River, which is a salmon river.

VENTILATION

The mine long section (see Figure 2) illustrates the present ventilation network in the B Zone. 4.7 m^3/s of air currently exhausts to the open pit.

POTENTIAL ZONE FOR BACTERIAL LEACHING

Two promising blocks of ore are the B-1 Zone crown pillar and the 8600 level sill pillar. These are indicated on the long section of the mine.

The B-1 Zone crown pillar contains 920,000 tonnes of material grading 1.97% Pb, 1.46% Cu, 3.97% Zn and 84.34 g/t Ag. This pillar consists of ore grade material plus other sulphides. A sample geological section through the crown pillar is shown in Figure 3. The 8600 level sill pillar contains 578,000 tonnes grading 1.59% Pb, 1.19% Cu, 3.69% Zn and 62.40 g/t Ag. These figures include ore grade material only.

The total amount of material in mine pillars is 5,170,950 tonnes grading 1.56%Pb, 1.06% Cu, 4.27% Zn and 60.34 g/t Ag.

TABLE 4

ANALYSIS OF MINE WATER - HEATH STEELE

(avg. values for 1982 except where noted)

	Cu mg/l	Zn mg/l	Fe mg/l	pH
B-4 PUMP STATION	94.62	320.41	-	3.1
6 LEVEL	2.94	241.36	-	4.0
TOTAL B-MINE	27.74	262.75	1157.1[1]	-

---

[1] Average of 12 samples tested in the period 10 February 1983 to 25 April 1983

TABLE 5

PUMPING VOLUMES - AVERAGE OF 2 YEARS[1]

HEATH STEELE

(1/day)

MONTH	B-4 PUMP STATION	6 LEV.PUMP STATION	TOTAL MINE
JANUARY	24,813	38,687	63,500
FEBRUARY	12,662	40,474	53,136
MARCH	13,200	45,826	59,026
APRIL	27,781	55,710	83,491
MAY	23,776	52,091	75,867
JUNE	16,096	41,208	57,304
JULY	13,605	36,242	49,847
AUGUST	13,105	42,836	55,941
SEPTEMBER	12,814	40,697	53,511
OCTOBER	21,134	44,766	65,900
NOVEMBER	19,041	50,092	69,133
DECEMBER	27,334	51,035	78,369
TOTAL	225,361	539,664	765,025

[1] 1981 AND 1982

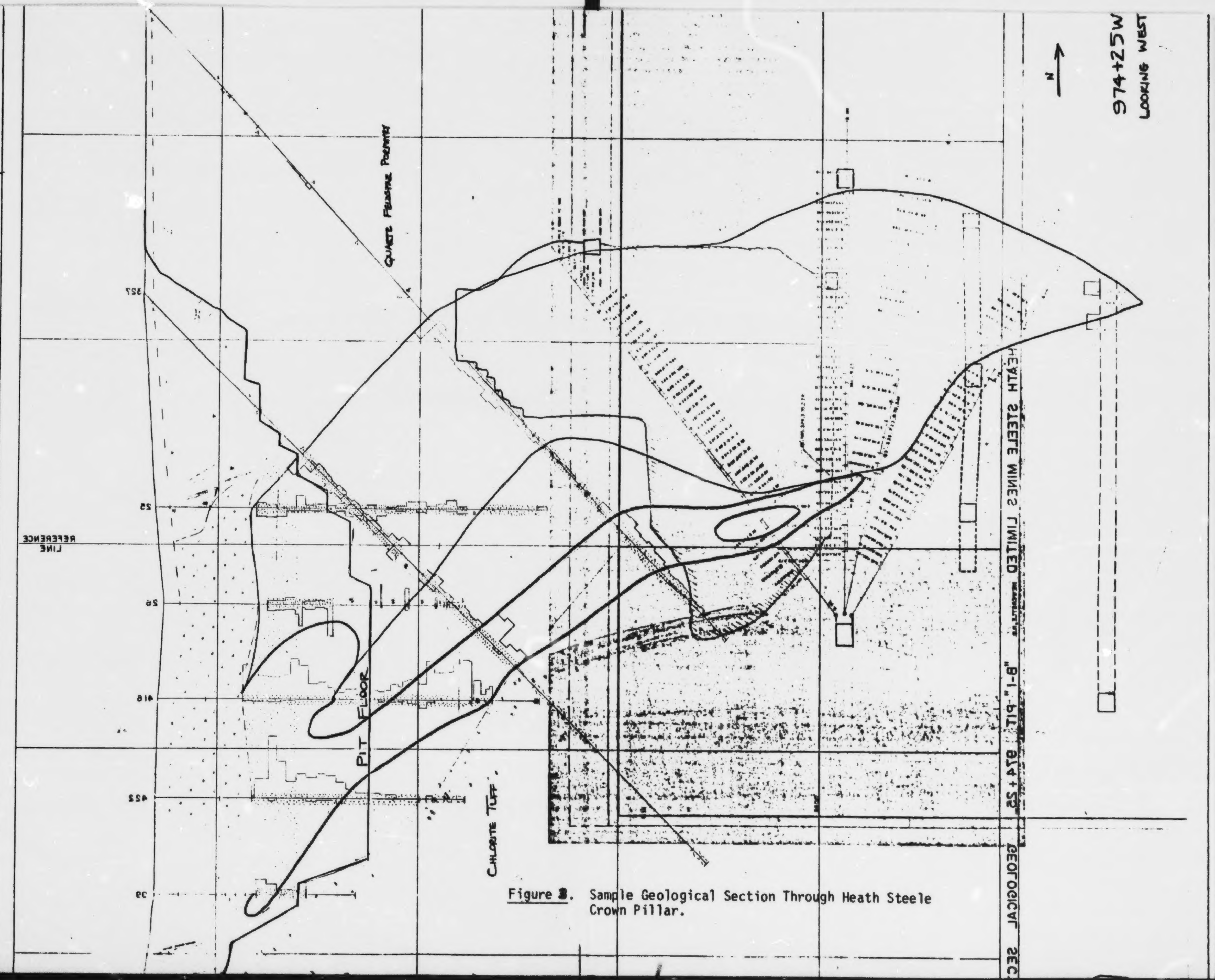

974+25W
LOOKING WEST
N
QUARTZ FELDSPAR PORPHYRY
REFERENCE LINE
327
25
26
418
422
30
PIT FLOOR
CHLORITE TUFF
HEATH STEELE MINES LIMITED
GEOLOGICAL SEC 25+974 "PIT "B-1"

Figure 3. Sample Geological Section Through Heath Steele Crown Pillar.

APPENDIX C

MINE SITE REPORT FOR
THE REMNOR PROJECT

REMNOR PROJECT

MINE SITE

Location

The Remnor Project refers to reserves of gold ore remaining in the Horne and Quemont mines in Noranda, Quebec. Noranda is approximately 550 km northwest of Montreal.

Topography

Gentle topography surrounds Noranda, and there are several lakes, rivers and marshes in the area. (see the topographic map in Figure 1). The mine site elevation is about 310 m, and the highest points of land near the mine are just over 360 m (see Figure 2).

Climatology

Tables 1 and 2 contain average monthly temperature and precipitation values for 1981 and 1983. Average annual precipitation for these two years was 842.4 mm. Some seasonal variation is evident, but rainfall is relatively high and fairly steady from May through October. Average temperature varies between -16.9°C (January) and 17.9°C (July). Absolute minimum and maximum values for these data are -37.3°C and 31.6°C respectively.

MINE CONFIGURATION

Orebodies

Remnor ore reserves are contained in both massive sulphide lenses and rhyolite zones. The long section in Figure 3 shows the large number of orebodies incorporated in the Remnor Project. These vary in size and configuration, but the overall volcanic structure hosting them strikes east-west, is irregular in dip and plunges to the east.

Figure 4 shows a detailed long section through the Remnor ore zones between surface and the 12th level. These ore zones range in size from 25,000 to 330,000 tonnes.

Mining Method

Production from the Horne mine was by blasthole and cut and fill mining methods as well as some shrinkage stoping.

Tonnage

363 tonnes per day of Remnor Project ore will be hoisted to the surface.

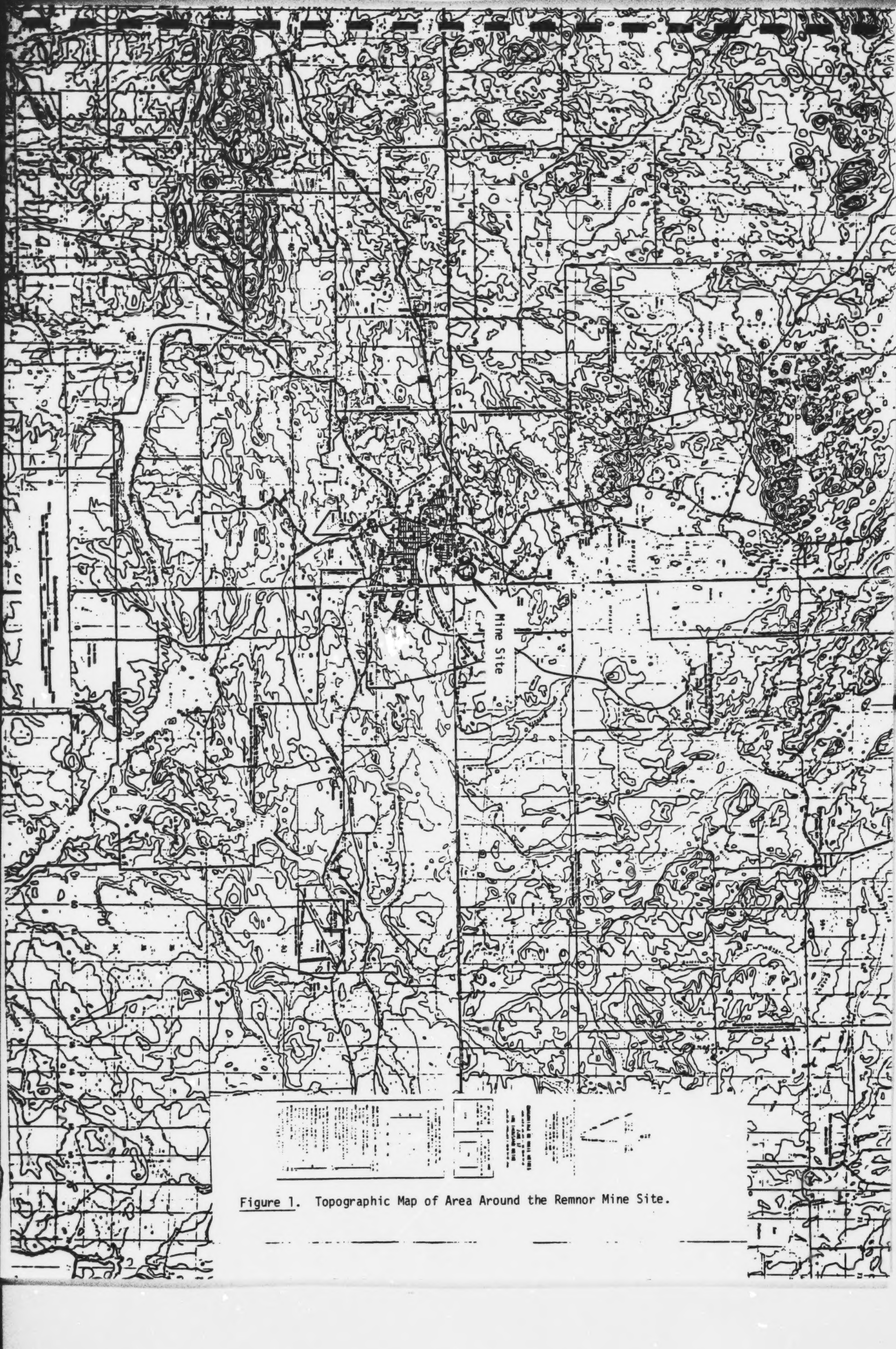


Figure 1. Topographic Map of Area Around the Remnor Mine Site.

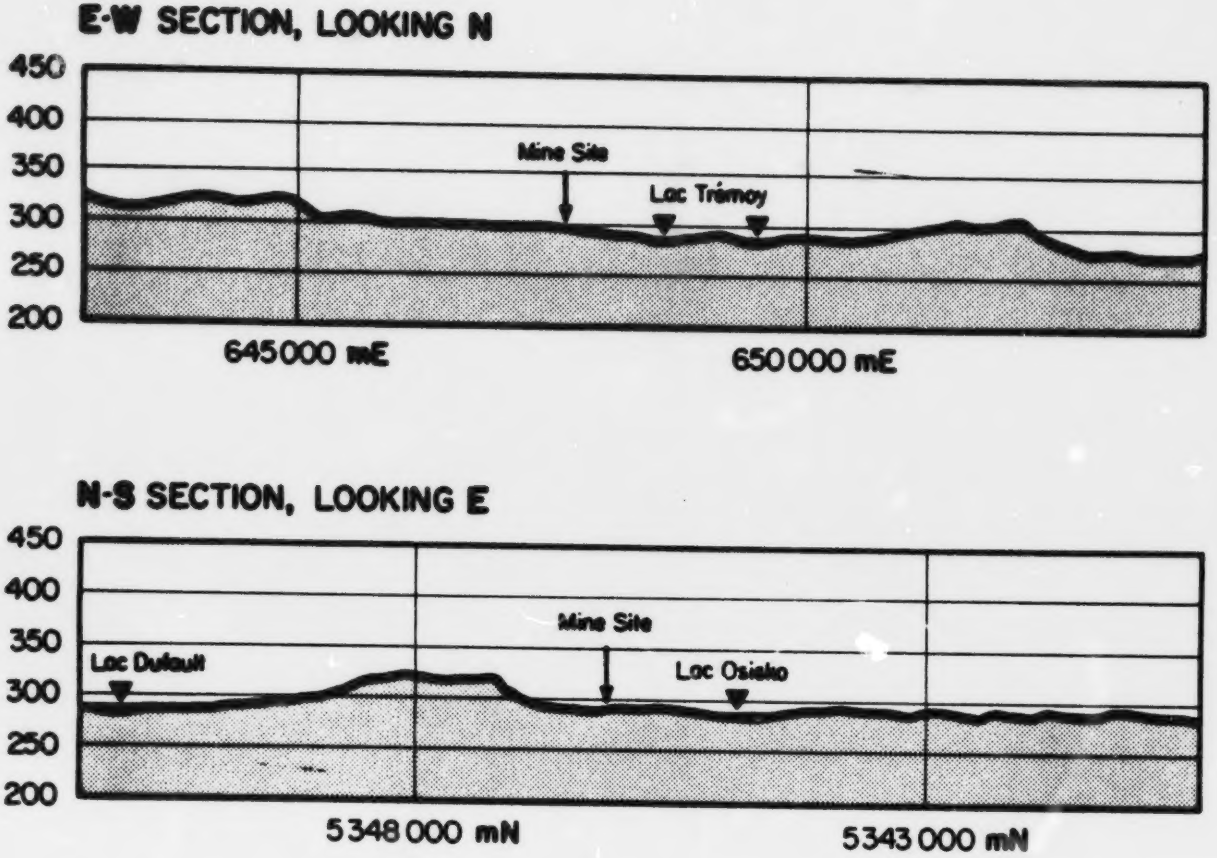


Figure 2. Topographic Sections Through the Remnor Mine Site.

TABLE 1

TEMPERATURE DATA - AVERAGE OF 2 YEARS[1]
REMNOR PROJECT[2]
(°C)

MONTH	AVERAGE TEMP.	AVERAGE		ABSOLUTE MAXIMUM	ABSOLUTE MINIMUM
		MAXIMUM	MINIMUM		
JANUARY	-16.9	-10.9	-22.9	1.5	-37.3
FEBRUARY	-11.2	- 4.4	-17.9	4.5	-37.0
MARCH	- 6.6	- 0.6	-12.6	8.5	-29.4
APRIL	0.9	6.2	- 4.5	17.5	-15.1
MAY	7.2	13.9	1.5	27.6	- 8.3
JUNE	14.7	21.3	8.1	31.4	0.9
JULY	17.9	25.0	10.7	31.6	3.1
AUGUST	17.1	24.0	10.1	29.7	2.8
SEPTEMBER	10.9	16.2	5.7	27.4	- 2.8
OCTOBER	3.6	8.7	- 1.6	19.5	-10.9
NOVEMBER	- 1.9	1.8	- 6.2	13.6	-16.9
DECEMBER	-13.6	- 8.7	-18.5	0.5	-34.0

[1] 1981 and 1983

[2] DATA FROM ROUYN AIRPORT

TABLE 2

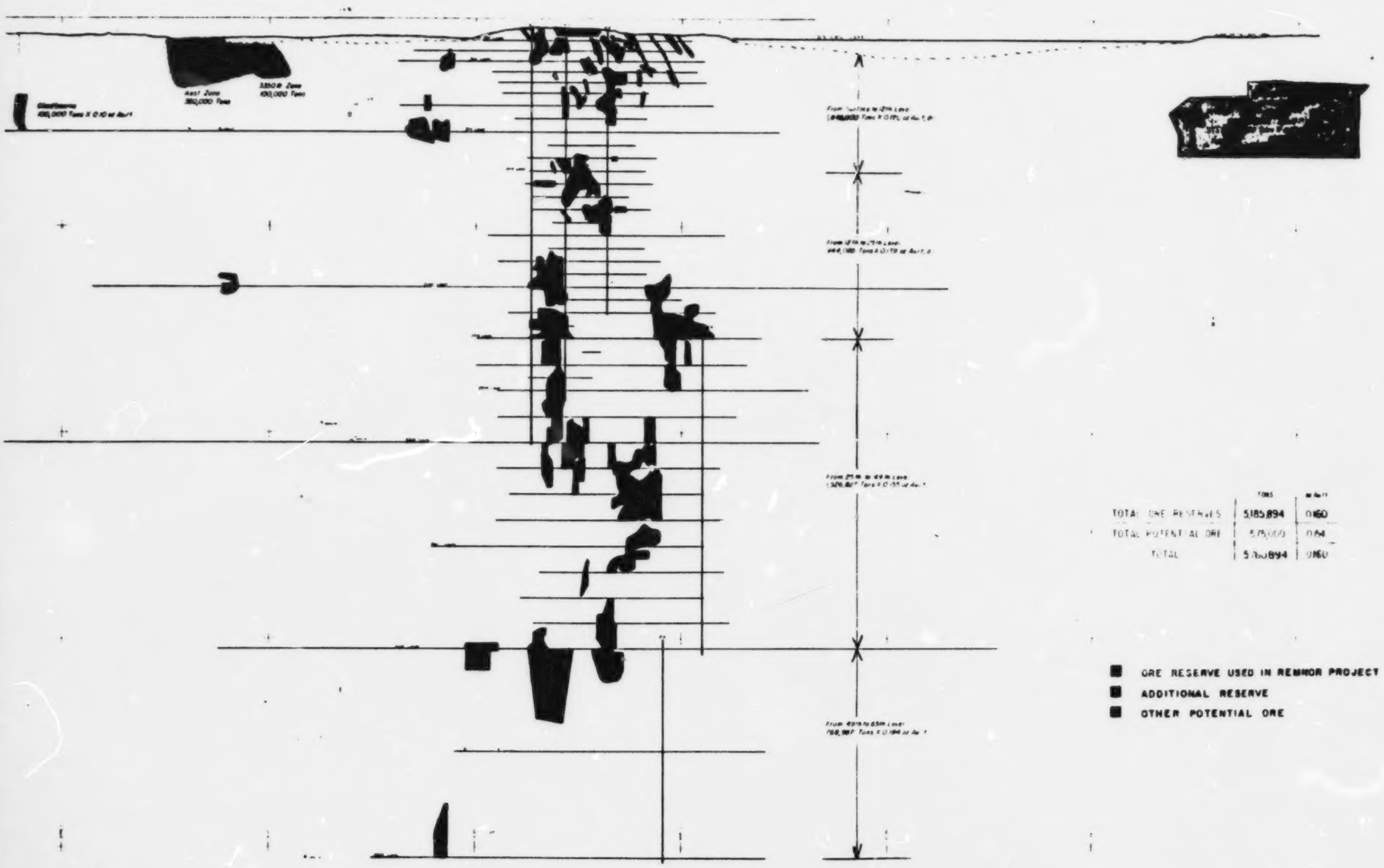
PRECIPITATION DATA - AVERAGE OF 2 YEARS[1]

REMNOR PROJECT[2]

MONTH	RAIN mm	SNOW cm	TOTAL PRECIP. mm
JANUARY	1.7	36.3	38.0
FEBRUARY	12.9	48.7	61.6
MARCH	21.2	34.6	55.8
APRIL	53.1	37.1	90.2
MAY	89.8	1.6	91.4
JUNE	63.9	0.0	63.9
JULY	76.7	0.0	76.7
AUGUST	65.7	0.0	65.7
SEPTEMBER	74.5	0.7	75.2
OCTOBER	77.4	13.1	90.5
NOVEMBER	24.9	38.0	62.9
DECEMBER	0.1	70.4	70.5
TOTAL	561.9	280.5	842.4

[1] 1981 and 1983

[2] DATA FROM VAL D'OR AIRPORT


Glenhaven
400,000 Tons X 0.10 oz Au/t
East Zone
350,000 Tons
3350 ft Zone
400,000 Tons
From Surface to 12th Level
[illegible] Tons X 0.175 oz Au/t
From 12th to 25th Level
944,088 Tons X 0.179 oz Au/t
From 25th to 49th Level
1,326,887 Tons X 0.159 oz Au/t
From 49th to 65th Level
768,987 Tons X 0.184 oz Au/t
ORE RESERVE USED IN REMNOR PROJECT
ADDITIONAL RESERVE
OTHER POTENTIAL ORE

	TONS	oz Au/t
TOTAL ORE RESERVES	5,185,894	0.160
TOTAL POTENTIAL ORE	575,000	0.164
TOTAL	5,760,894	0.160

Figure 3. Long Section of Remnor Project Looking North.

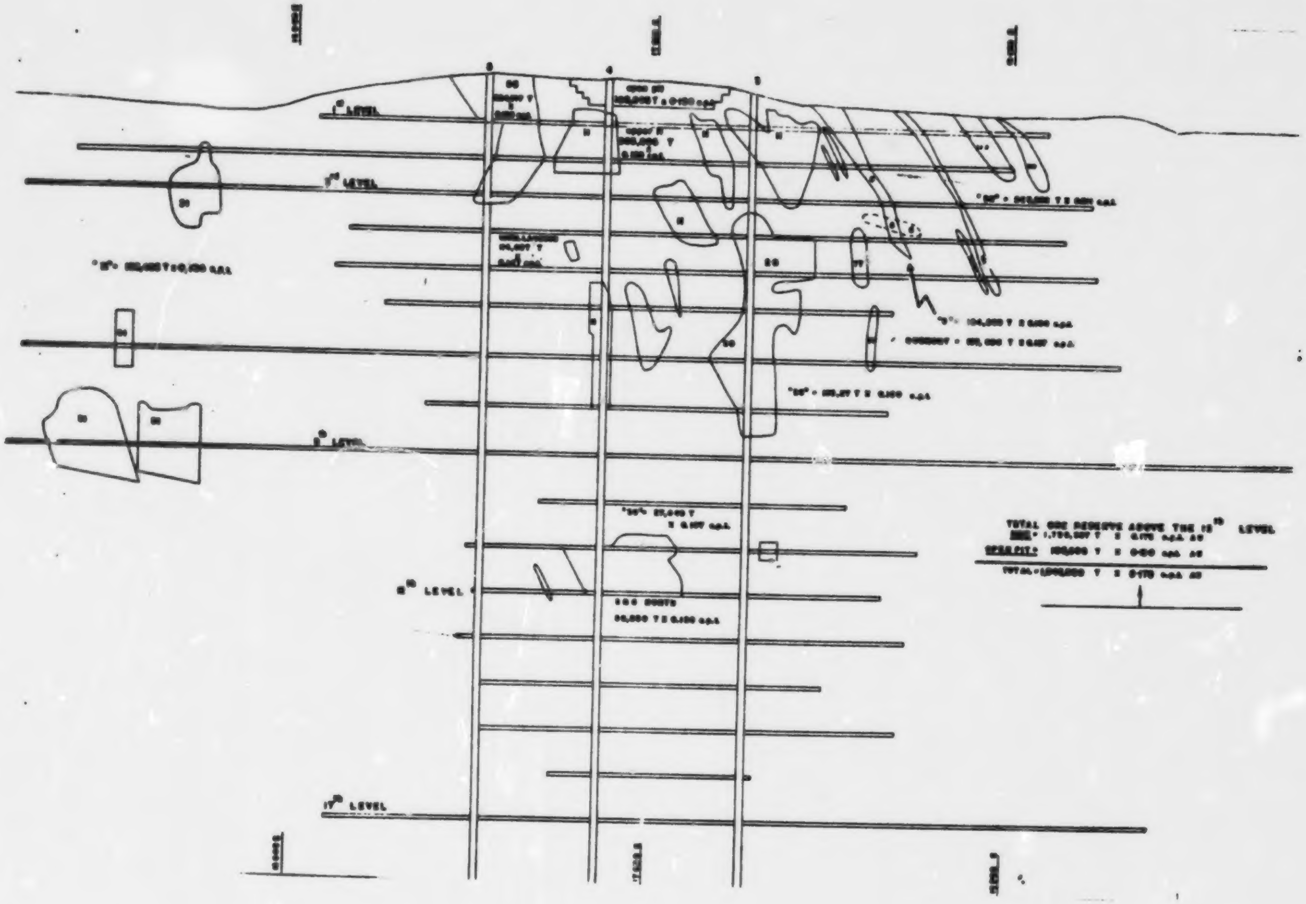

TOTAL ORE RESERVES ABOVE THE 12th LEVEL
OPEN PIT
12th LEVEL

Figure 4. Long Section Through Remnor Ore Zones Between Surface and 12th Level.

Surface Facilities

The surface facilities on the property consist of a mill, smelter and several shops and office buildings.

GEOLOGY

Mineralogy

Massive Sulphides:	These zones contain 50 to 70 pct sulphides comprising pyrite with minor associated tellurium, pyrrhotite and chalcopyrite.
Rhyolite :	These zones contain between 5 and 20 pct pyrite with minor associated tellurium.

Chemical Analysis

Chemical analyses of the massive sulphide and rhyolite ores are presented in Table 3.

Structure

The regional geology of the Horne and Quemont area is dominated by the Horne Creek Fault and the Andesite Fault, which intersect to form a wedge of the Horne Rhyolite Series that host the Horne orebodies (see Figure 5). The Horne Creek Fault strikes northeast dipping vertical to 75° southeast, and is up to 46 m wide in places. The Andesite Fault strikes east to southeast dipping vertically to steeply north.

The Horne Rhyolite Series have been folded as part of an east-striking anticline that plunges 45° to 65° to the east with an axial plane that dips from 70° to 80° to the south.

Local faulting underground at the Horne mine has been observed as displacement adjacent to many of the massive sulphide lenses. The most prominent trend is a northeast-striking, steeply dipping shearing. This shearing is strongly developed around the Upper H orebody and in the 55 zone. A northwest-striking set of subvertical fractures is also present but is not as prominent.

MINE WATER

During production, an average of 20,440 litres per day were pumped from the Horne mine. Table 4 contains average values for mine water analyses conducted during 1984.

VENTILATION

Total ventilation for the Remnor Project will be 105 m^3/s.

TABLE 3

CHEMICAL ANALYSIS OF REMNOR ORE

		RHYOLITE	MASSIVE SULPHIDE
Cu	%	0.52	0.55
Fe	%	10.90	46.60
Au	g/t	7.78	4.77
Ag	g/t	8.43	12.69
Mn	%	0.014	0.04
Co	%	0.01	0.02
In	g/t	1.47	27.43
Pb	%	0.0005	0.05
Zn	%	0.03	0.11
Ni	%	0.002	0.004
S	%	10.41	38.43
Sb	%	0.0057	0.01
Bi	%	0.0027	---
Cd	%	0.001	0.024
Se	%	0.0061	negligible
Te	%	0.0014	negligible
As	%	<0.01	<0.01
Hg	%	negligible	0.08
SiO_2	%	62.60	6.10
Al_2O_3	%	9.22	1.70
FeS_2	%	18.16	---
K_2O	%	2.45	---
CaO	%	0.22	---
Na_2O	%	0.32	---
MgO	%	0.53	---

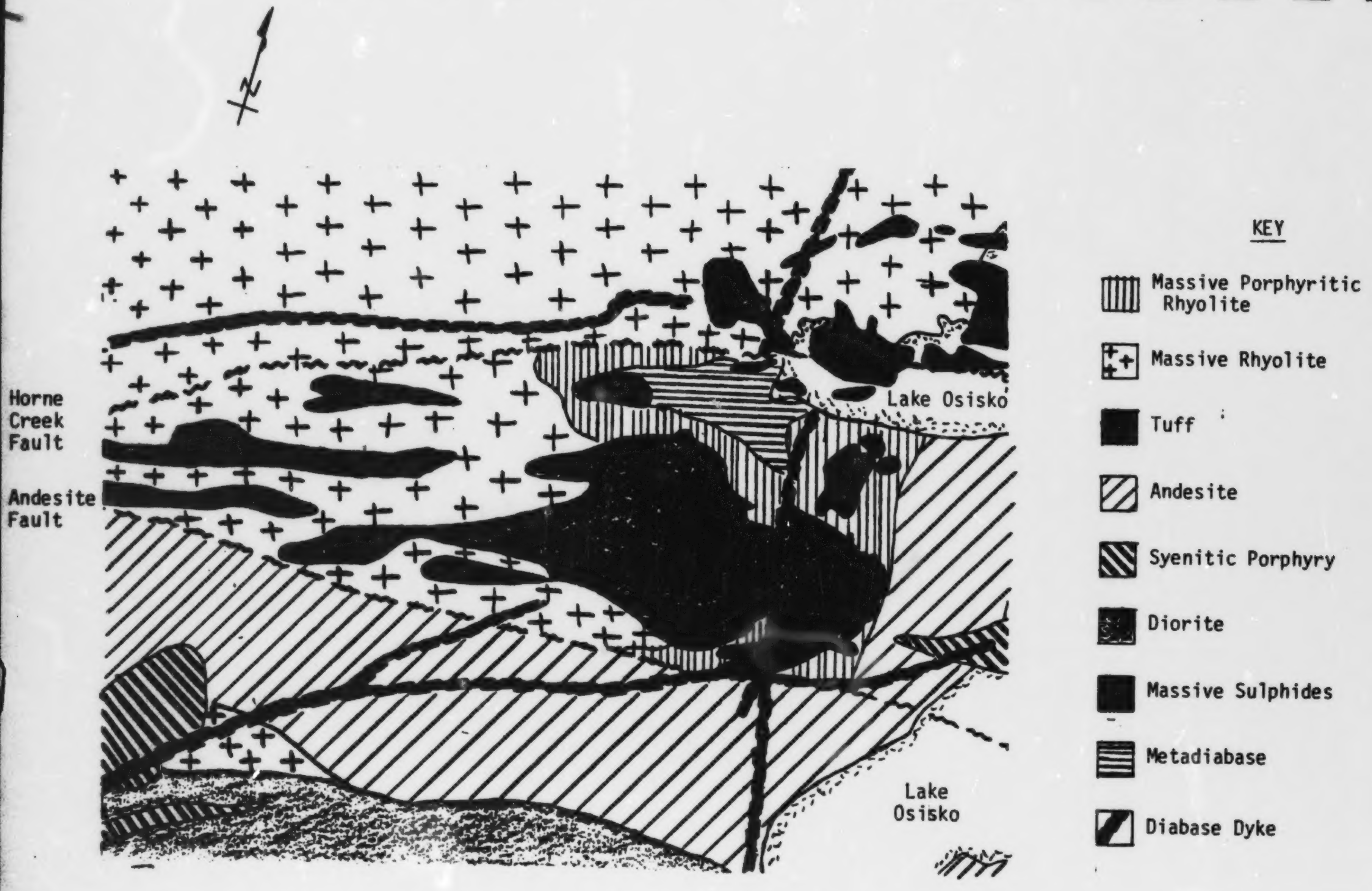

N
Horne Creek Fault
Andesite Fault
Lake Osisko
Lake Osisko
KEY
Massive Porphyritic Rhyolite
Massive Rhyolite
Tuff
Andesite
Syenitic Porphyry
Diorite
Massive Sulphides
Metadiabase
Diabase Dyke

Figure 5. Geology of the Horne and Quemont Area.

TABLE 4

ANALYSIS OF MINE WATER - REMNOR
(averages of readings taken in 1984)

Cu	mg/l	26.63
Zn	mg/l	400.53
Cd	mg/l	506.25
Fe	mg/l	1486.14
Hg	µg/l	< 0.03
pH		3.0

POTENTIAL ZONES FOR BACTERIAL LEACHING

Of the ore reserves outlined in detail above the 12th level, two zones have relatively high copper values - the Upper H zone (massive sulphide) and the 31 zone (rhyolite).

In the Upper H zone, 331,850 tonnes of proven ore grading 4.73 g/t Au and 0.56% Cu are contained in 17 separate blocks. The 31 zone, consisting of 6 blocks, contains 123,800 tonnes of proven ore grading 4.66 g/t Au and 0.52% Cu.

Future potential material for leaching consists of the 5 zone at depth, which contains 50,112,000 tonnes grading 1.30 g/t Au, 0.10% Cu, 0.59% Zn, 21.0% Fe and 41.8% FeS_2. The "high pyrite zone" within 5 zone contains 18,448,000 tonnes grading 1.41 g/t Au, 0.15% Cu, 1.15% Zn, 31.8% Fe and 65.0% FeS_2.

APPENDIX D

MINE SITE REPORT FOR THE GOLDSTREAM MINE

MACLAREN FOREST PRODUCTS INC.

GOLDSTREAM MINING DIVISION

MINE SITE

Location

The Goldstream copper-zinc property is located in the Goldstream River valley, 93 km north of Revelstoke in the Selkirk mountains of southeastern British Columbia.

Topography

The valley floor elevation at the mine site is about 650 m, and the mountains on each side top at about 2000 m. Figure 1 illustrates the local topography, and Figure 2 shows north-south and east-west sections through the mine site.

Climatology

Statistics for the mine site are not available, but data for Mica Dam (137 km north of Goldstream) and Revelstoke are shown in Tables 1 and 2.

MINE CONFIGURATION

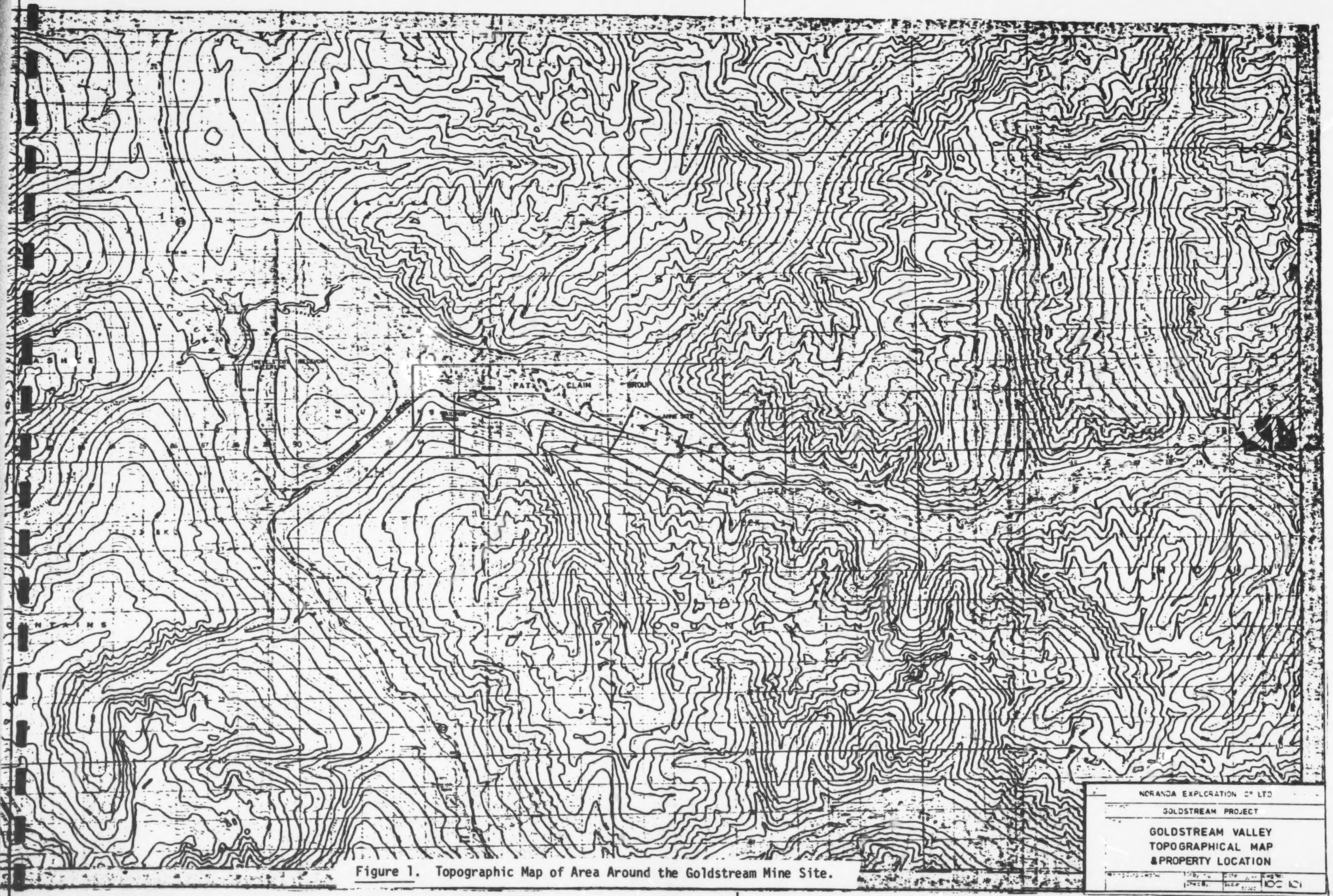
Orebody

The Goldstream deposit lies within a sequence of metamorphosed and deformed sedimentary and volcanic rocks. The shape is that of a "flattened rod" or "ruler". The deposit consists of a single continuous bed of massive and disseminated sulphides varying in thickness from 1 to 15 m. The average thickness is 3 m.

The orebody outcrops at 920 m elevation on the south side of the Goldstream River valley. Strike length varies from 340 m at outcrop to 180 m at depth. Average dip is 33°, and the orebody rakes at 45° off true dip. Both hangingwall and footwall contacts roll on dip and on strike. The deposit is open down dip, but proven down dip length is 1050 m.

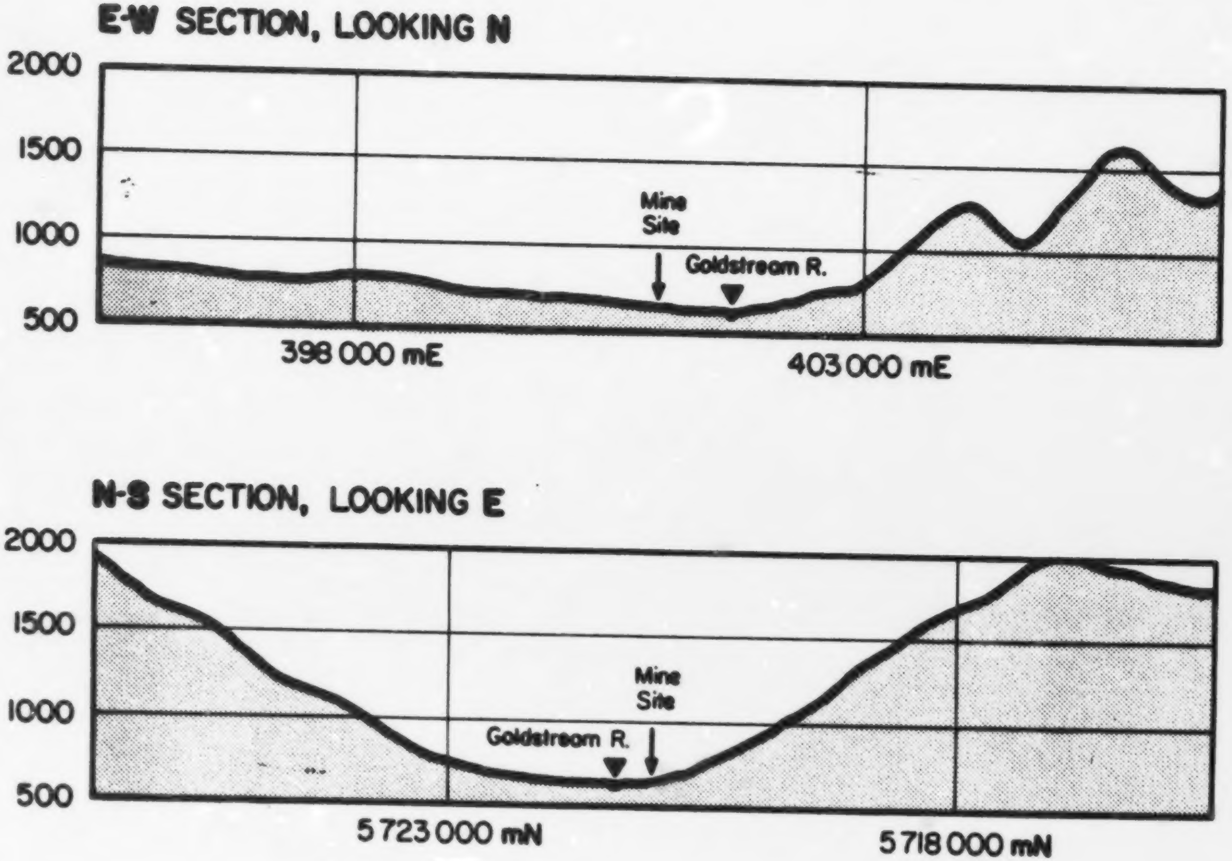
Mining Method

The mining method used at Goldstream is called "Step Room and Fill". Stopes are mined from the upper end to the lower end by a series of consecutive cuts (see Figure 3). Each stope is delineated by sill pillars and rib pillars. Following completion of a block, mining of the sill and rib pillars proceeds from the bottom to the top of the block. Backfilling is carried out prior to the mining of each pillar.



PAT CLAIM GROUP
MINE SITE
TREE FARM LICENCE
NORANDA EXPLORATION Cº LTD
GOLDSTREAM PROJECT
GOLDSTREAM VALLEY
TOPOGRAPHICAL MAP
&PROPERTY LOCATION
100 101

Figure 1. Topographic Map of Area Around the Goldstream Mine Site.


E-W SECTION, LOOKING N

N-S SECTION, LOOKING E

Figure 2. Topographic Sections Through the Goldstream Mine Site.

TABLE 1

TEMPERATURE DATA

MICA DAM AND REVELSTOKE[1]
(°C)

MONTH	MICA DAM (7 YEARS OF RECORD)			REVELSTOKE (69 YEARS OF RECORD)		
	MEAN DAILY	EXTREME MAXIMUM	EXTREME MINIMUM	MEAN DAILY	EXTREME MAXIMUM	EXTREME MINIMUM
JANUARY	-10.4	5.0	-33.9	- 5.9	10.6	-34.4
FEBRUARY	- 5.3	6.7	-23.3	- 1.8	12.8	-32.2
MARCH	- 1.3	12.8	-23.3	1.9	22.2	-22.2
APRIL	3.5	18.3	- 8.3	7.2	28.3	-15.0
MAY	9.3	27.8	- 3.9	12.8	37.2	- 6.7
JUNE	14.5	35.6	0.0	16.4	36.7	- 1.7
JULY	17.1	36.1	3.3	19.4	40.6	0.6
AUGUST	15.8	35.0	2.8	18.2	38.3	- 6.1
SEPTEMBER	11.4	28.9	- 2.8	13.7	37.2	- 6.7
OCTOBER	4.9	18.3	- 5.6	7.0	25.6	-12.2
NOVEMBER	- 2.1	9.4	-19.4	0.8	15.0	-22.8
DECEMBER	- 7.2	3.3	-37.2	- 3.6	17.2	-32.8

---

[1] Mica Dam elevation 580 m
Revelstoke elevation 456 m

TABLE 2

PRECIPITATION DATA
MICA DAM AND REVELSTOKE[1]

MONTH	MICA DAM (7 YEARS OF RECORD)		REVELSTOKE (69 YEARS OF RECORD)	
	TOTAL mm	SNOW cm	TOTAL mm	SNOW cm
JANUARY	205.7	193.5	147.3	125.5
FEBRUARY	152.4	120.4	104.1	79.0
MARCH	129.5	78.7	73.7	26.7
APRIL	83.8	8.4	50.8	2.5
MAY	55.9	Trace	53.3	Trace
JUNE	61.0	0.0	71.1	0.0
JULY	55.9	0.0	53.3	0.0
AUGUST	73.7	0.0	55.9	0.0
SEPTEMBER	68.6	3.6	71.1	0.0
OCTOBER	154.9	9.7	101.6	3.3
NOVEMBER	185.4	114.3	121.9	48.8
DECEMBER	228.6	207.8	144.8	108.0
TOTAL	1455.4	736.4	1048.9	393.8

---

[1] Mica Dam elevation 580 m
Revelstoke elevation 456 m

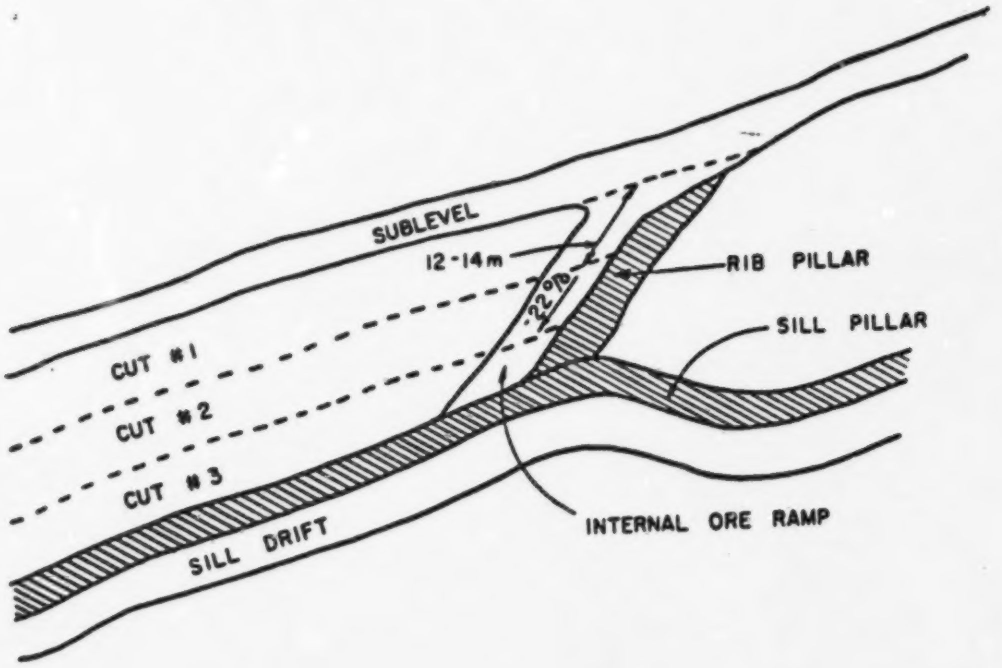

SUBLEVEL
12-14m
RIB PILLAR
22%
SILL PILLAR
CUT #1
CUT #2
CUT #3
SILL DRIFT
INTERNAL ORE RAMP

Figure 3. Cut Sequence for a 'Step Room and Fill' Stope, After Humphrey, 1983.

Tonnage

Prior to shutdown in April, 1984, the mine was producing 1360 tonnes per day.

Surface Facilities

The surface facilities, as shown in Figure 4, include a concentrator, a service building, a cold storage area and a carpenter shop.

GEOLOGY

Mineralogy

A north-south vertical section through the Goldstream deposit is shown in Figure 5. Figure 6 illustrates the situation of rock units within the stratigraphic column.

The ore zone is in the massive sulphide layer, which is composed of intimately intermixed pyrrhotite, chalcopyrite and sphalerite with numerous subrounded inclusions of quartz, phyllite and carbonate.

The massive sulphide layer is enclosed in an envelope of siliceous chlorite and sericite phyllite grading to fine-grained sericite-chlorite quartzite. Pyrrhotite, chalcopyrite and minor sphalerite are disseminated in this material, increasing substantially near the sulphide layer.

The garnet zone which overlies the sericite phyllite contains abundant spessartine garnets. This zone includes layers of chert, chloritic phyllite and chlorite-rich graphitic and calcareous quartzite. Pyrrhotite is present in thin irregular laminations or in discontinuous streaks, and grunerite occurs in some siliceous layers.

Chemical Analysis

Table 3 contains average values for analyses of selected samples of the massive sulphide, hangingwall and footwall.

Structure

The mine site is located on the limb of a fold plunging to the north east. Folding is seen on a regional scale, due to two major phases of deformation in the past, but is not prevalent on a mine scale.

Three major regional joint sets exist which are approximately orthogonal. Orientation of these joint sets is 090/48N, 203/73SE and 120/68W respectively, corresponding with mine north. Mine north is about 20° east of true north. The first joint set has associated foliation, while shearing is associated with the second and third sets. Shearing is most prominent with the second set.

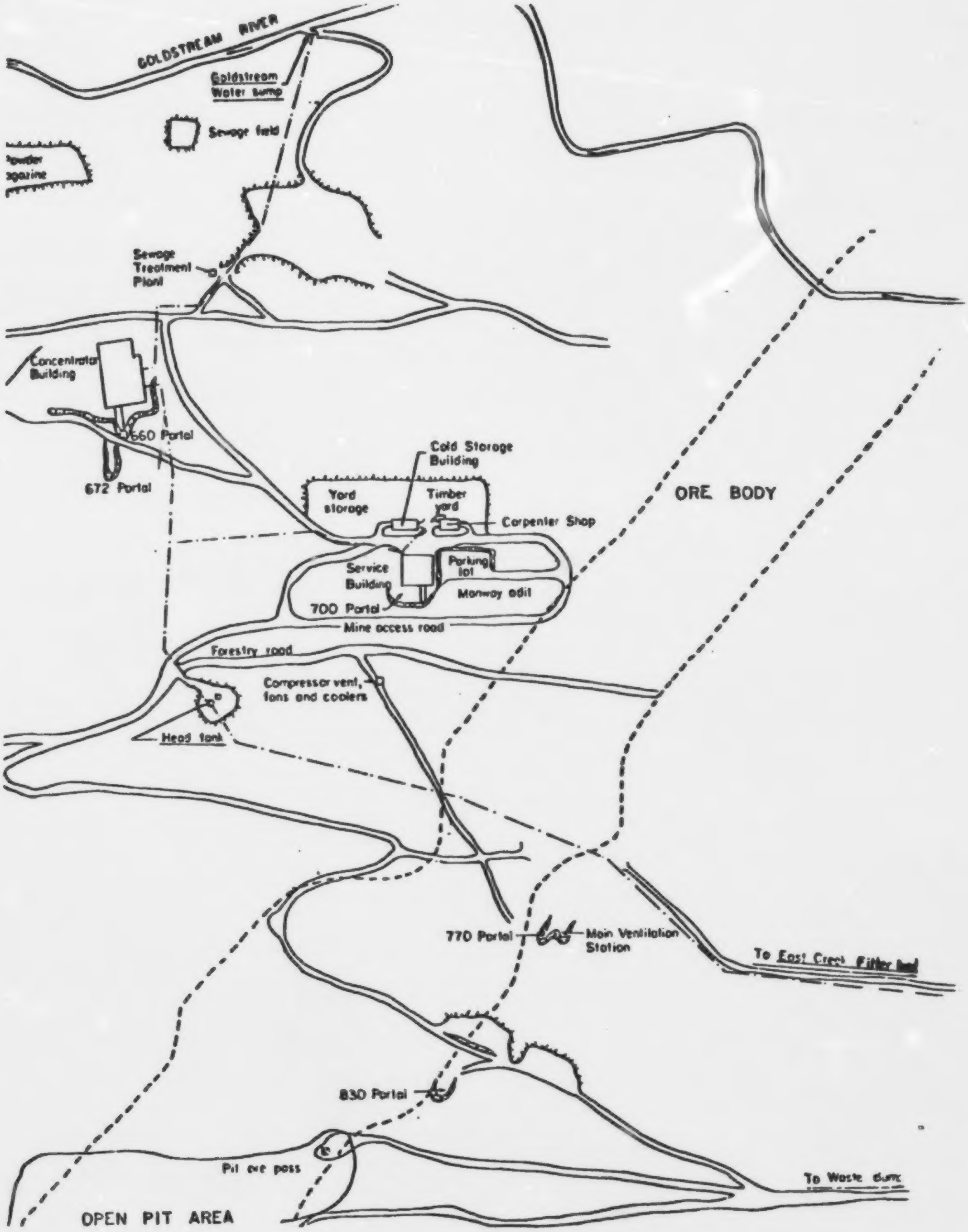
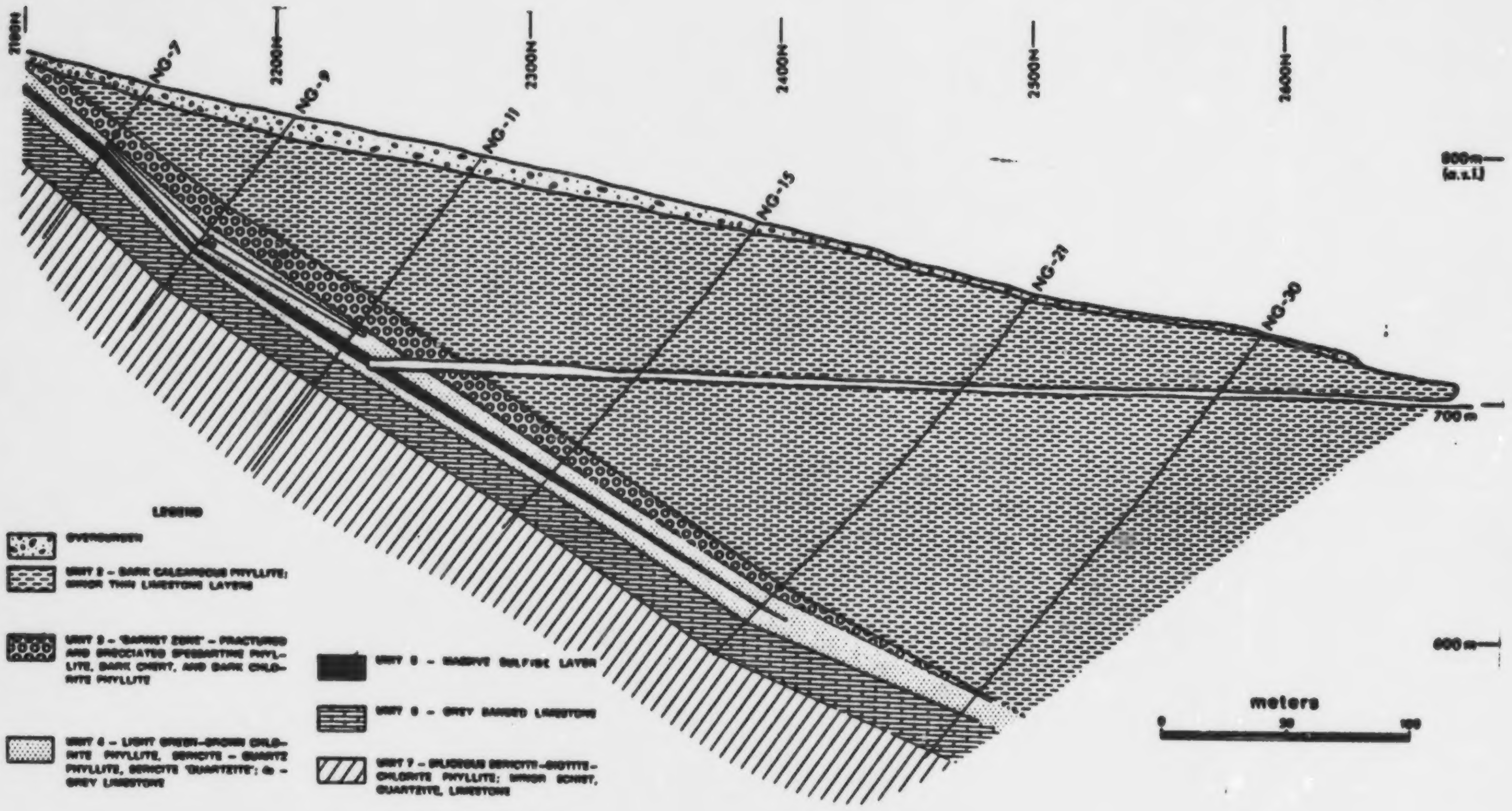

GOLDSTREAM RIVER
Goldstream Water sump
Sewage field
owder
agazine
Sewage Treatment Plant
Concentrator Building
660 Portal
672 Portal
Cold Storage Building
Yard storage
Timber yard
Carpenter Shop
ORE BODY
Service Building
Parking lot
Manway adit
700 Portal
Mine access road
Forestry road
Compressor vent, fans and coolers
Head tank
770 Portal
Main Ventilation Station
To East Creek Filter bed
830 Portal
Pit ore pass
To Waste dump
OPEN PIT AREA

Figure 4. Surface Layout of Goldstream Property, After Humphrey, 1983.



2100N
2200N
2300N
2400N
2500N
2600N
NG-7
NG-9
NG-11
NG-15
NG-21
NG-30
800m
(a.s.l.)
700 m
600 m
meters
0
50
100
LEGEND
OVERBURDEN
UNIT 2 – DARK CALCAREOUS PHYLLITE; MINOR THIN LIMESTONE LAYERS
UNIT 3 – 'GARNET ZONE' – FRACTURED AND BRECCIATED SPESSARTINE PHYLLITE, DARK CHERT, AND DARK CHLORITE PHYLLITE
UNIT 4 – LIGHT GREEN-BROWN CHLORITE PHYLLITE, SERICITE – QUARTZ PHYLLITE, SERICITE 'QUARTZITE'; 4a – GREY LIMESTONE
UNIT 5 – MASSIVE SULFIDE LAYER
UNIT 6 – GREY BANDED LIMESTONE
UNIT 7 – SILICEOUS SERICITE–BIOTITE–CHLORITE PHYLLITE; MINOR SCHIST, QUARTZITE, LIMESTONE

Figure 5. North-South Vertical Section Through the Goldstream Deposit, Looking West, After Høy et al., 1984.

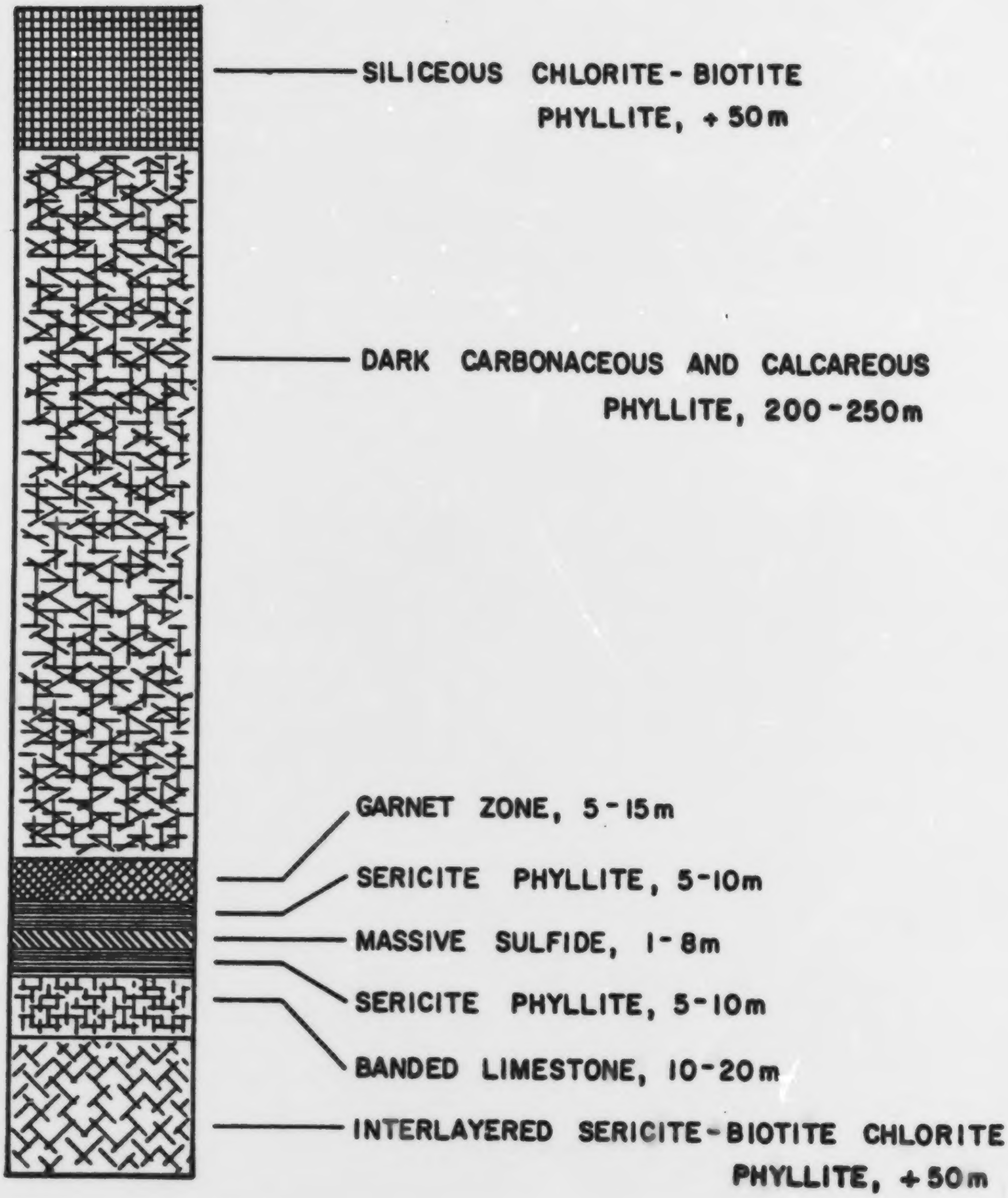

SILICEOUS CHLORITE-BIOTITE PHYLLITE, +50m
DARK CARBONACEOUS AND CALCAREOUS PHYLLITE, 200-250m
GARNET ZONE, 5-15m
SERICITE PHYLLITE, 5-10m
MASSIVE SULFIDE, 1-8m
SERICITE PHYLLITE, 5-10m
BANDED LIMESTONE, 10-20m
INTERLAYERED SERICITE-BIOTITE CHLORITE PHYLLITE, +50m

Figure 6. Goldstream Rock Units, After Humphrey, 1983.

TABLE 3

CHEMICAL ANALYSES OF GOLDSTREAM ORE

		SULPHIDE	HANGING WALL	FOOTWALL
Cu	%	3.53	1.29	0.41
Zn		4.18%	0.31%	348.89 ppm
Pb		0.47%	174.14 ppm	79.22 ppm
Fe	%	25.05	4.51	6.50
Ag	ppm	20.08	4.64	3.35
Au	ppm	0.08	--	--
Ni	ppm	69.32	29.43	41.60
Co	ppm	71.95	18.57	21.40
Mn	%	0.31	--	--
SiO_2	%	--	60.66	61.97
Al_2O_3	%	--	7.06	11.34
Fe_2O_3	%	--	8.69	11.39
MgO	%	--	3.67	1.49
CaO	%	--	6.97	2.92
Na_2O	%	--	0.84	0.63
K_2O	%	--	1.02	2.67
TiO_2	%	--	0.45	0.81
MnO	%	--	0.19	0.17

Fault zones exist 15 to 20 m above and below the ore zone and parallel to it. In general, ground conditions at Goldstream are good and support problems are local in nature and confined to the vicinity of faults or shears.

Ambient rock temperature is estimated to be between 10°C and 15°C.

MINE WATER

Pumping volume has not been measured at Goldstream, but is estimated to be between 6.3 and 9.5 l/s. Spring run-off and heavy rain do not noticeably affect pumping volumes. Table 4 contains average values for mine water analyses conducted between 1982 and 1984 inclusive.

VENTILATION

A total of 113 m^3/s of fresh air is supplied to the underground workings.

POTENTIAL ZONES FOR BACTERIAL LEACHING

Since the mine is currently shut down for economic reasons, the whole deposit offers potential material for leaching. Table 5 details the remaining reserves.

TABLE 4

ANALYSIS OF MINE WATER - GOLDSTREAM
(averages of readings taken 1982-84)

pH	7.9
Dissolved Metals	
As	0.41 ppm
Cu	0.12 ppm
Fe	0.49 ppm
Pb	0.07 ppm
Zn	0.09 ppm
SO_4	105.80 ppm
Total Metals	
Susp.solids	10846 ppm
As	1.26 ppm
Cu	88.29 ppm
Fe	445.25 ppm
Pb	7.14 ppm
Zn	63.71 ppm

TABLE 5

REMAINING RESERVES - GOLDSTREAM

	TONNES	% Cu	% Zn
Developed Ore to 655 m. Elev.			
Pit (to 860 m elev)	100,000	3.23	1.84
830 - 860	179,758	3.89	1.93
770 - 830	333,724	3.64	2.29
715 - 770	285,899	4.39	3.87
655 - 715	267,696	3.93	3.00
Total Developed	1,167,077	3.89	2.75
Drill Indicated Ore to 400 m. Elev.			
655 - 400	2,007,285	3.29	2.35
Total Developed & Drill Indicated			
Pit to 400 m. elev.	3,174,362	3.51	2.50

APPENDIX E

MINE SITE REPORT FOR THE GECO MINE

NORANDA MINES LIMITED

GECO DIVISION

MINE SITE

Location

The Geco mine is in the Manitouwadge mining camp in northwestern Ontario, 320 km east of Thunder Bay.

Topography

Figure 1 illustrates the topography in the area, which is moderately hilly. Mine site elevation is about 335 m, and the highest hilltops locally are at 442 m elevation (see Figure 2).

Climatology

Tables 1 and 2 contain average monthly temperature and precipitation values for 1980 and 1981. Average annual precipitation for these two years was 737.9 mm. Relatively high rainfall is recorded from June through September. Average temperature varies between -15.9°C (January) and 18.2°C (July) for these data. Absolute minimum and maximum values recorded were -37.3°C and 30.8°C respectively.

MINE CONFIGURATION

Orebodies

The regional geological structure consists of a broad easterly plunging synform. The Geco orebody follows a dragfold in the south limb of this synform. It is one vertical, lenticular, continuous zone of mineralization which is interrupted by several diabase dykes and is offset by the Fox Creek Fault. The average horizontal length on any level is approximately 730 m and the average width is about 20 m. The bottom of the orebody plunges to the east at 35°. Figure 3 shows a long section of the Geco mine.

Mining Method

Bulk blasthole mining has accounted for 95 pct of the ore extracted at Geco to date. Primary stopes were 21 m wide, 91 m high transverse slices at 37 and 46 m intervals along strike. These have all been mined and filled with waste rock fill. Hydraulically placed 30:1 cemented mill tailings were used to stabilize the walls of these stopes, and the pillars between them are now being mined.

The remaining 5 pct of extraction is by cut and fill mining, which is used where the ore is too narrow for longitudinal blasthole stoping and more ground support is required.

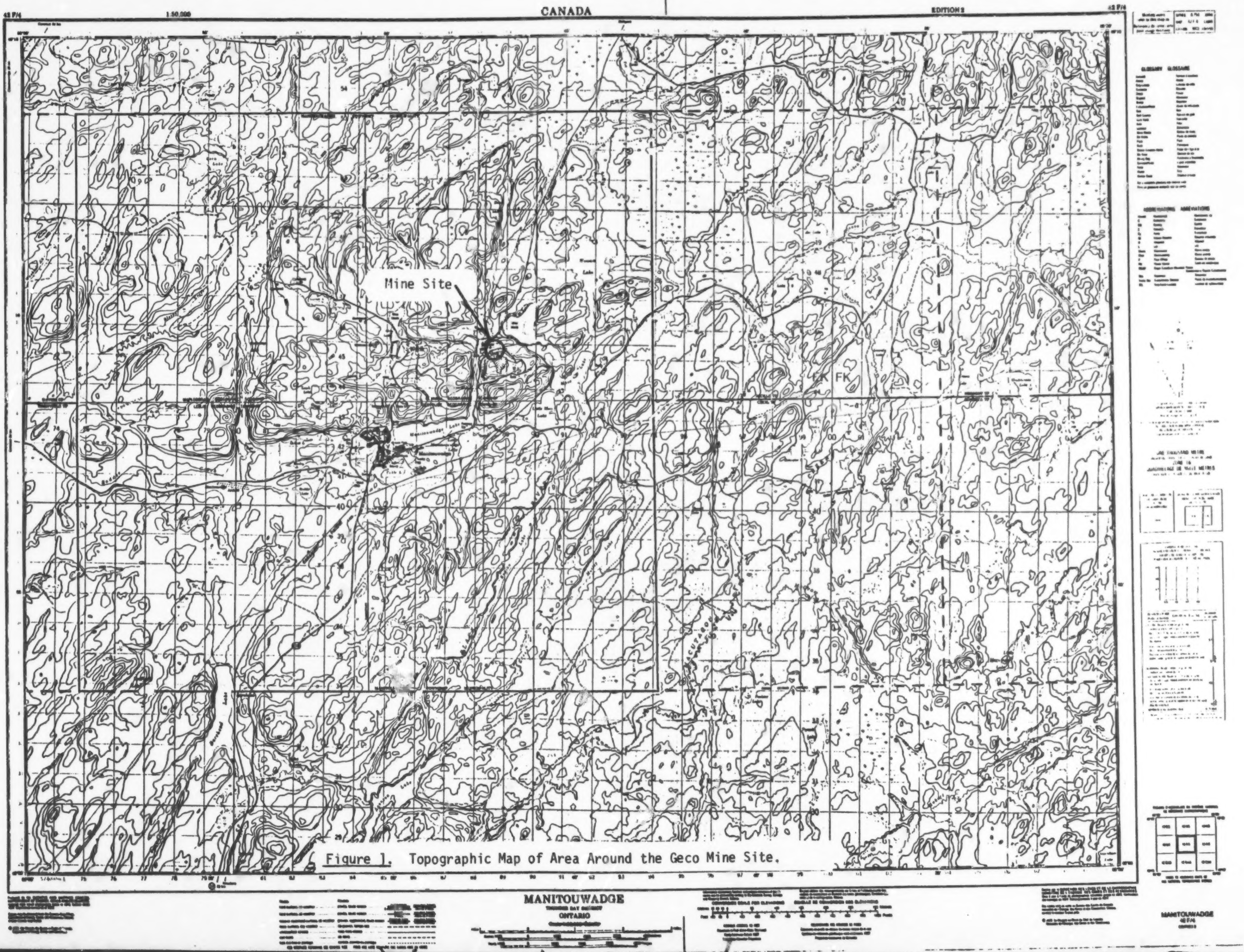

43 F/4
1:50 000
CANADA
EDITION 2
Mine Site
MANITOUWADGE
ONTARIO

Figure 1. Topographic Map of Area Around the Geco Mine Site.

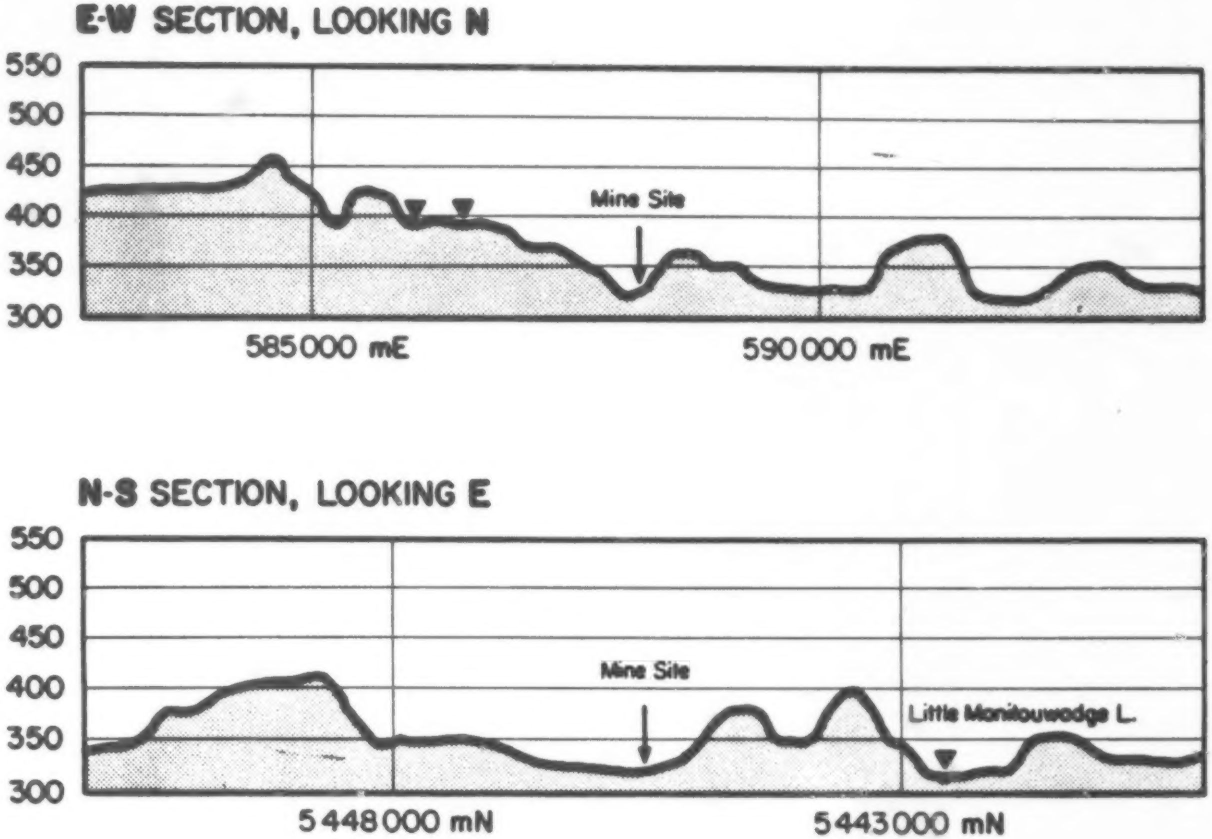


Figure 2. Topographic Sections Through the Geco Mine Site.

TABLE 1

TEMPERATURE DATA - AVERAGE OF 2 YEARS[1]

GECO MINE

(°C)

MONTH	AVERAGE TEMP.	AVERAGE		ABSOLUTE MAXIMUM	ABSOLUTE MINIMUM
		MAXIMUM	MINIMUM		
JANUARY	-15.9	-10.9	-21.0	1.8	-37.3
FEBRUARY	-13.4	- 7.8	-19.0	3.3	-35.0
MARCH	- 6.6	- 0.8	-12.4	10.8	-27.8
APRIL	1.5	7.5	- 4.7	19.3	-16.0
MAY	9.9	17.1	2.8	30.3	- 6.5
JUNE	12.9	19.2	6.7	30.8	- 1.5
JULY	18.2	25.6	10.8	30.5	4.0
AUGUST	16.8	23.0	10.8	29.3	4.0
SEPTEMBER	8.7	13.2	4.3	24.0	- 5.8
OCTOBER	2.8	8.1	- 2.6	15.5	-11.3
NOVEMBER	- 2.9	1.6	- 7.4	9.3	-20.0
DECEMBER	-14.1	- 9.6	-18.7	0.3	-33.5

[1] 1980 and 1981

TABLE 2

PRECIPITATION DATA - AVERAGE OF 2 YEARS[1]
GECO MINE

MONTH	RAIN mm	SNOW cm	TOTAL PRECIP. mm
JANUARY	0.8	49.0	49.8
FEBRUARY	18.0	32.5	50.5
MARCH	5.2	31.0	36.2
APRIL	23.7	32.8	56.5
MAY	47.8	3.5	51.3
JUNE	94.6	0.0	94.6
JULY	68.2	0.0	68.2
AUGUST	73.6	0.0	73.6
SEPTEMBER	101.7	1.0	102.7
OCTOBER	41.5	16.6	58.1
NOVEMBER	4.2	31.2	35.4
DECEMBER	0.0	61.0	61.0
TOTAL	479.3	258.6	737.9

[1] 1980 and 1981

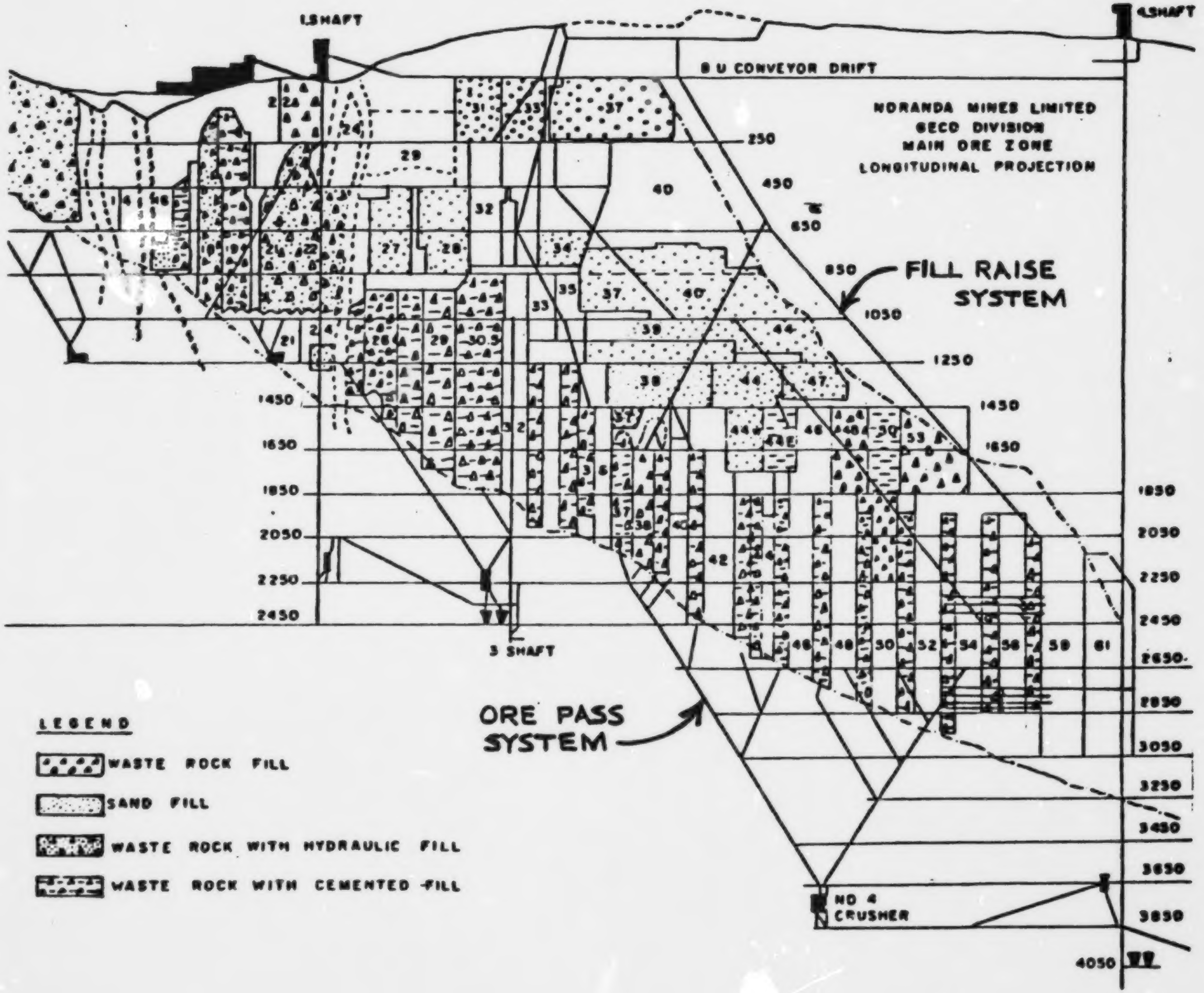

1 SHAFT
4 SHAFT
8 U CONVEYOR DRIFT
NORANDA MINES LIMITED
GECO DIVISION
MAIN ORE ZONE
LONGITUDINAL PROJECTION
FILL RAISE SYSTEM
ORE PASS SYSTEM
3 SHAFT
NO 4 CRUSHER
250
450
650
850
1050
1250
1450
1650
1850
2050
2250
2450
2650
2850
3050
3250
3450
3650
3850
4050
LEGEND
WASTE ROCK FILL
SAND FILL
WASTE ROCK WITH HYDRAULIC FILL
WASTE ROCK WITH CEMENTED FILL

Figure 3. Long Section of the Geco Mine.

Tonnage

The Geco mine has a production capacity of 3630 tonnes per day.

Surface Facilities

Geco's surface facilities consist of a mill which produces copper, lead and zinc concentrates by flotation; and offices, warehouses and shops. Figures 4 and 5 illustrate surface layout of facilities around No. 1 and No. 4 shafts respectively.

GEOLOGY

Mineralogy

Figure 6 illustrates the relative positions of ore zones and host rocks at Geco. Figure 7 shows the location of ore zones within the stratigraphic column.

The Sericite Schist Group, which ranges up to 200 m in width, contains all the known orebodies at Geco. The 4/2 Copper Zone is composed of fine to medium grained chalcopyrite - pyrrhotite - pyrite contained in a highly siliceous matrix. The sulphides are present in both massive and disseminated concentrations.

The Main Orebody consists of a core of massive coarse-grained sulphides containing 10 - 60 pct pyrite, 5 - 30 pct sphalerite, 2 - 25 pct chalcopyrite and minor galena and silver. A zone of disseminated sulphides in sericite schist forms an enveloupe around this core.

The 8/2 Zinc Zone contains fine to coarse grained sphalerite, pyrite and silver.

Chemical Analysis

Table 3 contains the element compositions of the main rock types.

Structure

Folding : Geco deposits have undergone several phases of folding. The dominant pattern at the mine is a set of Z-shaped northeast trending asymmetrical folds. Numerous shear zones were developed parallel to the axial planes of this fold system.

Faulting : Two regional faults affect the mine area. The vertical Fox Creek Fault, which strikes north-south and is 15-30 m wide, affects mining on the western end of the deposit. On the eastern end of the property, the vertical Mose Lake Fault strikes northwesterly and averages 10 m in width.

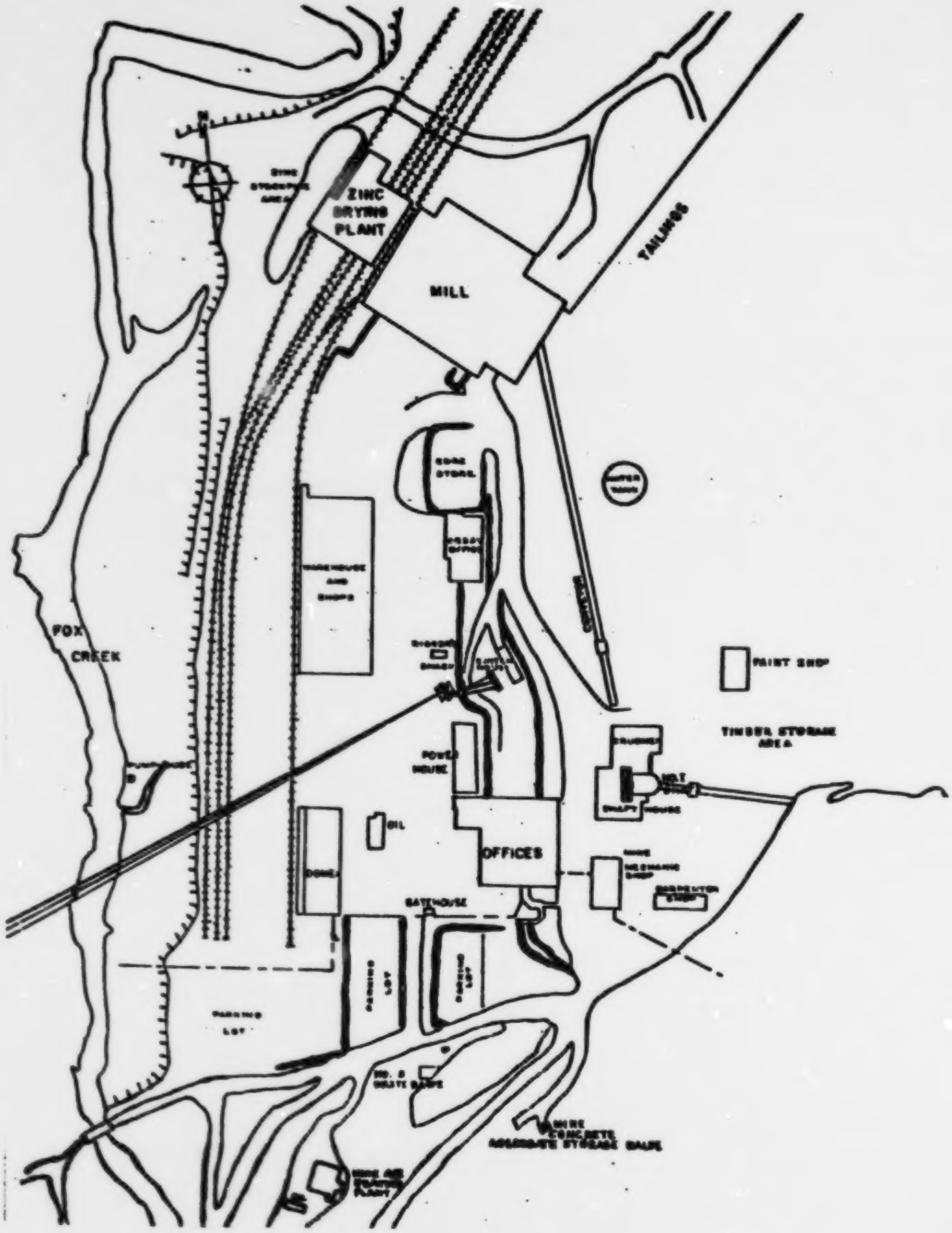

ZINC STOCKPILE AREA
ZINC DRYING PLANT
MILL
TAILINGS
CORE STORE
WATER TANK
ASSAY OFFICE
WAREHOUSE AND SHOPS
FOX CREEK
PAINT SHOP
TIMBER STORAGE AREA
CRUSHER
NO. 1 SHAFT
SHAFT HOUSE
POWER HOUSE
OIL
OFFICES
MINE MECHANIC SHOP
CARPENTER SHOP
BATHHOUSE
PARKING LOT
PARKING LOT
PARKING LOT
NO. 2 WASTE DUMP
MINE CONCRETE AGGREGATE STORAGE DUMP
MINE AIR HEATING PLANT

Figure 4. Surface Layout of No. 1 Shaft Area, Geco Mine.

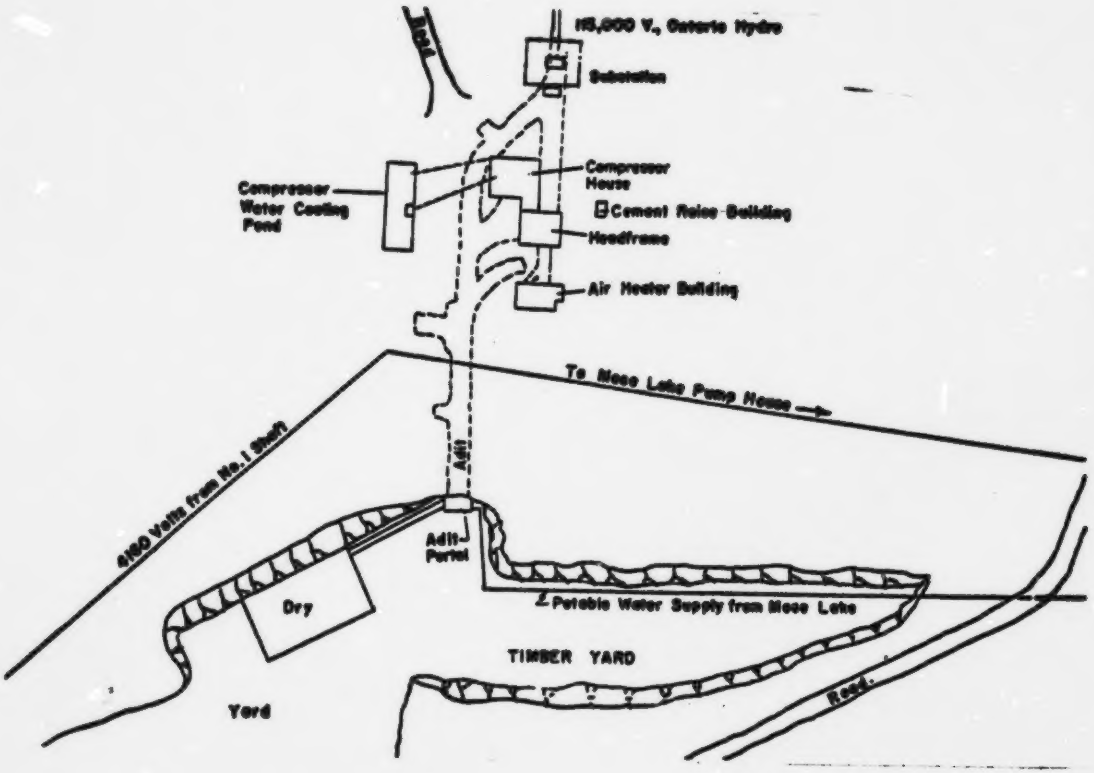

115,000 V., Ontario Hydro
Road
Substation
Compressor Water Cooling Pond
Compressor House
Cement Raise Building
Headframe
Air Heater Building
To Mose Lake Pump House
4160 Volts from No. 1 Shaft
Adit
Adit Portal
Dry
Potable Water Supply from Mose Lake
TIMBER YARD
Yard
Road

Figure 5. Surface Layout of No. 4 Shaft Area, Geco Mine.

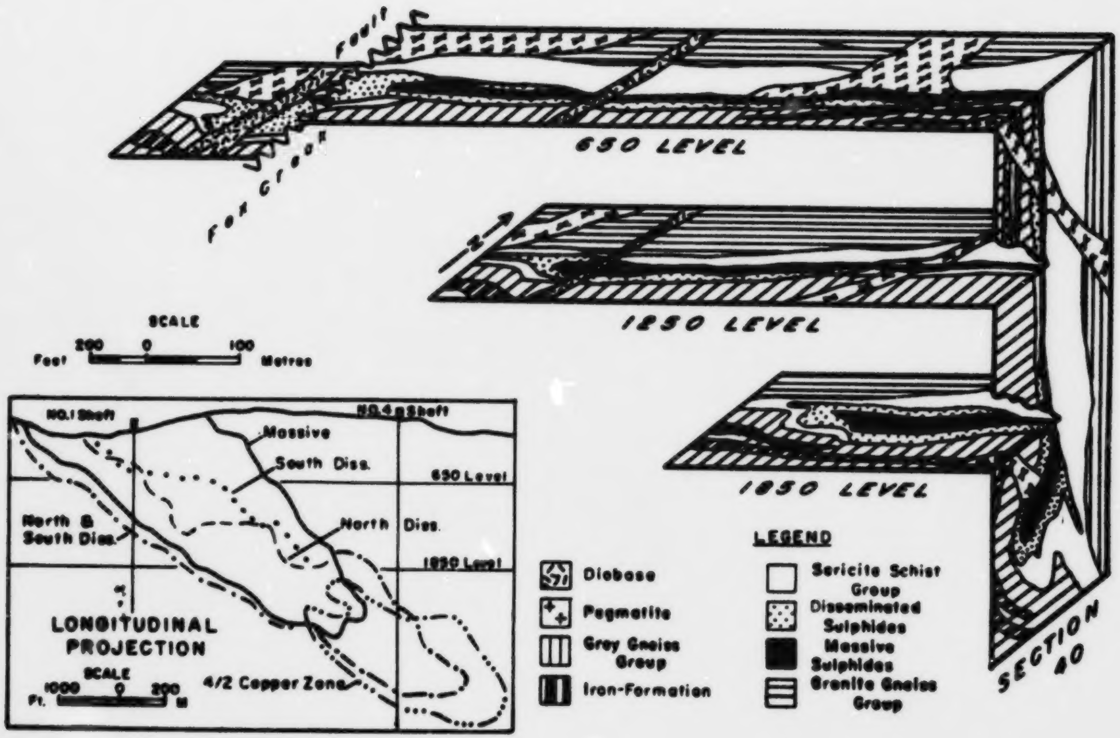

Fault
Fox Creek
650 LEVEL
1250 LEVEL
1850 LEVEL
SECTION 40
SCALE
Feet 200 0 100 Metres
NO.1 Shaft
NO.4 Shaft
Massive
South Diss.
650 Level
North & South Diss.
North Diss.
1850 Level
LONGITUDINAL PROJECTION
SCALE
Ft. 1000 0 200 M
4/2 Copper Zone
Diabase
Pegmatite
Grey Gneiss Group
Iron-Formation
LEGEND
Sericite Schist Group
Disseminated Sulphides
Massive Sulphides
Granite Gneiss Group

Figure 6. Isometric Projection of West End of Geco Deposit After Friesen et al, 1982.

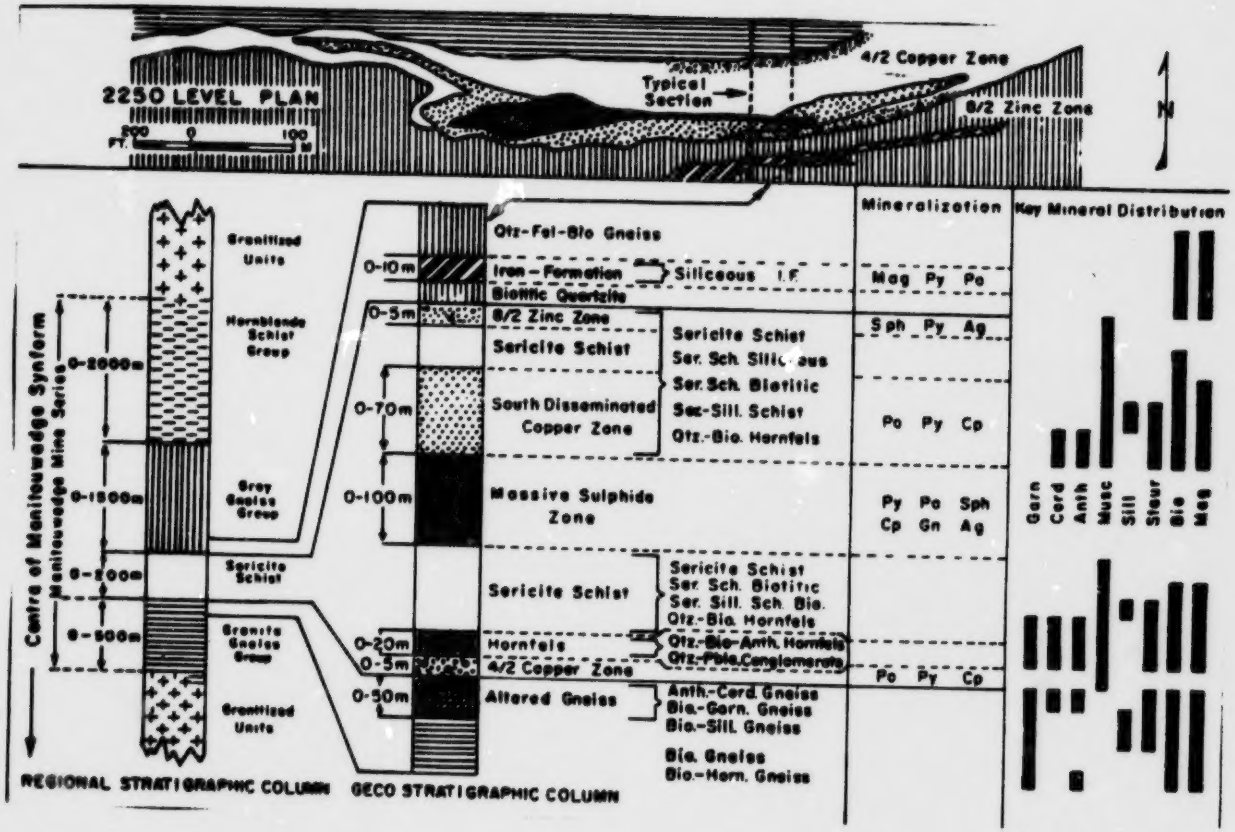

2250 LEVEL PLAN
200 FT.
0
100 M
Typical Section
4/2 Copper Zone
8/2 Zinc Zone
N
Centre of Manitouwadge Synform
Manitouwadge Mine Series
0-2000m
0-1500m
0-200m
0-500m
Granitized Units
Hornblende Schist Group
Grey Gneiss Group
Sericite Schist
Granite Gneiss Group
Granitized Units
REGIONAL STRATIGRAPHIC COLUMN
0-10m
0-5m
0-70m
0-100m
0-20m
0-5m
0-50m
Qtz-Fel-Bio Gneiss
Iron - Formation
Siliceous I.F.
Biotitic Quartzite
8/2 Zinc Zone
Sericite Schist
South Disseminated Copper Zone
Massive Sulphide Zone
Sericite Schist
Hornfels
4/2 Copper Zone
Altered Gneiss
Sericite Schist
Ser. Sch. Siliceous
Ser. Sch. Biotitic
Ser-Sill. Schist
Qtz.-Bio. Hornfels
Sericite Schist
Ser. Sch. Biotitic
Ser. Sill. Sch. Bio.
Qtz.-Bio. Hornfels
Qtz.-Bio-Anth. Hornfels
Qtz.-Pble.Conglomerate
Anth.-Cord. Gneiss
Bio.-Garn. Gneiss
Bio.-Sill. Gneiss
Bio. Gneiss
Bio.-Horn. Gneiss
GECO STRATIGRAPHIC COLUMN
Mineralization
Mag Py Po
Sph Py Ag
Po Py Cp
Py Po Sph
Cp Gn Ag
Po Py Cp
Key Mineral Distribution
Garn
Cord
Anth
Musc
Sill
Staur
Bio
Mag

Figure 7. Stratigraphic Column, Geco Mine. After Friesen et al., 1982.

TABLE 3

AVERAGE MAJOR ELEMENT COMPOSITIONS OF THE MAIN ROCK TYPES AT GECO (%)

ROCK TYPE	SiO_2	Al_2O_3	Total Fe	MgO	CaO	Na_2O	K_2O	TiO_2	P_2O_5	MnO	
Granite	70.57	15.17	1.51	0.52	1.78	4.51	3.49	0.27	0.05	0.02	
Qtz Diorite	61.54	15.57	5.58	3.16	4.23	3.37	2.42	0.65	0.26	0.10	
Hnb Schist	50.60	14.64	10.57	5.74	8.75	2.29	0.63	1.23	0.27	0.23	HORNBLENDE SCHIST GROUP
Qtz-Fel-Bio Gn	64.80	12.82	8.76	2.09	3.17	2.29	2.20	0.21	0.05	0.66	GREY GNEISS GROUP
Silicieous IF	67.43	0.40	24.76	1.42	1.80	0.00	0.03	0.04	0.00	0.54	
Chloritic IF	66.22	0.74	25.92	1.94	2.21	0.30	0.08	0.03	0.12	1.65	
Bio Quartzite	76.64	11.70	2.03	0.44	0.36	3.12	4.45	0.11	0.00	0.03	
Ser. Schist	84.53	7.58	1.77	0.37	0.19	0.32	1.74	0.09	0.01	0.01	SERICITE SCHIST GROUP
Ser.Schist Bio'c	71.23	10.56	6.05	2.79	0.31	0.73	3.17	0.13	0.01	0.04	
Ser.-Sill-Schist	78.67	9.80	3.93	1.98	0.19	0.24	1.63	0.11	0.00	0.02	
Qtz-Bio-Anth-Hfls	69.90	8.77	11.30	4.32	0.23	0.20	0.70	0.10	0.01	0.05	
Bio-Anth-Cord Gn	51.25	13.08	20.56	6.83	1.28	0.15	0.26	2.92	0.58	0.26	GRANITE GNEISS GROUP
Bio-Sill Gneiss	74.09	12.03	5.04	3.27	0.21	0.07	2.05	0.29	0.06	0.06	
Bio-Hnb Gneiss	45.53	15.88	16.11	6.71	6.39	1.43	0.78	2.41	0.33	0.41	

Rock temperature at the 3850 level east of No. 4 shaft is approximately 13°C.

MINE WATER

Table 4 contains an analysis of Geco mine water. The water is slightly acidic (pH = 4.5). Monthly water volumes pumped are shown in Table 5 for 1983. An average of 3,434,019 l/day of water is pumped. Average water temperature is between 10°C and 13°C.

VENTILATION

A total of 240 m^3/s of fresh air is supplied to the underground workings. Of this amount, about 39 m^3/s is supplied to lower levels of the mine in the No. 4 shaft area.

POTENTIAL ZONES FOR BACTERIAL LEACHING

No. 4 Shaft is situated on mine section 61 + 50. The current economic mining limit is on section 63 + 00, and at the 3250 level of the mine. The potential zone for leaching consists of all low-grade material east of this limit. An estimated 900,000 to 1,000,000 tonnes of sericite schist exists between sections 64 + 00 and 74 + 00. This material is well foliated and exhibits parallel jointing in line with the foliation. Figure 8 illustrates diamond drill information on section 67 + 00.

An existing drift on 2650 level extends eastward to between sections 64 + 00 and 65 + 00, and provides access to about 90,000 tonnes of this sub-ore grade material.

TABLE 4

ANALYSIS OF MINE WATER - GECO

(averages of readings taken in 1980)

Cu	ppm	7.7
Zn	ppm	229.1
Fe	ppm	190.1
pH		4.5

TABLE 5

1983 PUMPING VOLUMES - GECO

(1/day)

MONTH	1250 L No. 1 Station	1250 L No. 2 Station	2450 L	4050 L	TOTAL MINE
JANUARY		1,569,360		362,669	1,932,029
FEBRUARY		1,547,189		244,746	1,791,935
MARCH	131,880	1,711,127		254,966	2,097,973
APRIL	739,291	1,158,336		247,566	2,145,193
MAY	1,203,394	1,680,113		391,241	3,274,748
JUNE	2,180,397	2,132,701		499,674	4,812,772
JULY	1,595,733	2,212,267		435,201	4,243,201
AUGUST	876,993	1,559,469		395,636	2,832,098
SEPTEMBER	640,492	1,454,734	442,893	256,651	2,794,770
OCTOBER	1,058,329	1,737,504	771,490	252,767	3,820,090
NOVEMBER	861,938	2,234,907	1,529,685	377,027	5,003,557
DECEMBER	1,279,227	2,940,898	1,608,921	630,820	6,459,866
AVERAGE					3,434,019

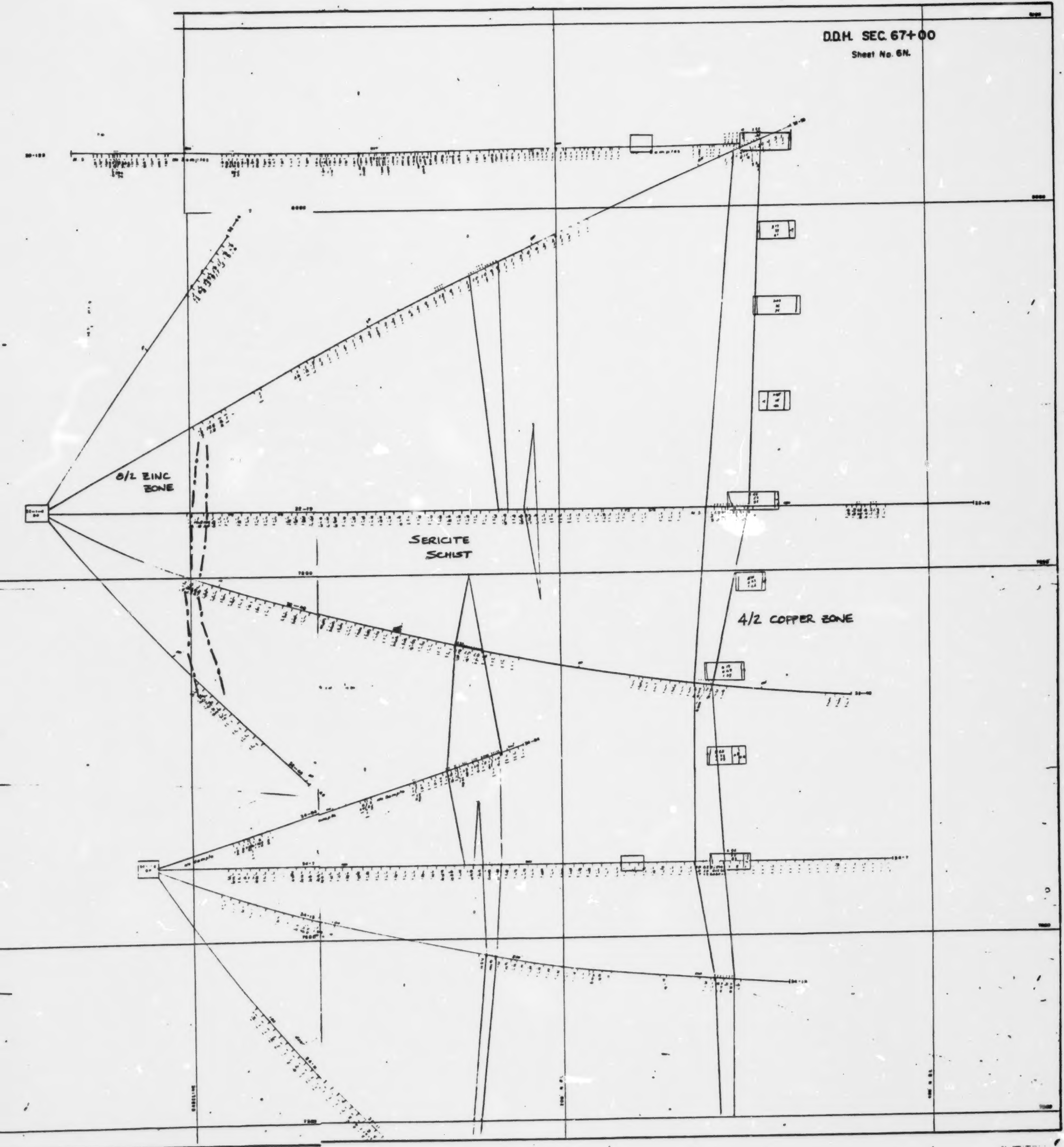

D.D.H. SEC. 67+00
Sheet No. 6N.
8/2 ZINC ZONE
SERICITE SCHIST
4/2 COPPER ZONE

Figure 8. Geological Section 67+00 of the Geco Mine.

APPENDIX F

MINE SITE REPORT FOR THE MATTABI MINE

MATTABI MINES LIMITED

MINE SITE

Location

The Mattabi mine is located in the Sturgeon Lake area of northwestern Ontario, about 100 km north of Ignace.

Topography

Relatively gentle topography surrounds the mine site, and there are several lakes in the area. Figure 1 is a topographic map of the area. North-south and east-west sections through the mine site are shown in Figure 2. The mine site elevation is about 427 m and the highest point locally is a hill southwest of the mine, at 495 m elevation.

Climatology

Detailed temperature and precipitation data are not recorded at the mine.

MINE CONFIGURATION

Orebody

The entire sequence of volcanic rocks containing the Mattabi orebody strikes east-west, dips to the north at 80° and plunges at 15° to 20° to the east. Figures 3 and 4 illustrate the geology of the mine area.

The orebody itself is 550 m long and 90 m wide at its widest point. It is divided into two zones separated by the mine cherty rhyolite tuffs. Each of these zones contains five distinctive "facies": (1) high sphalerite ore, (2) high pyrite, low sphalerite ore, (3) massive chalcopyrite ore, (4) high galena ore and (5) stringer ore and alteration pipe. Figure 5 is a long section of the mine.

Mining Method

Small diameter (2-1/8 inch) longhole open stoping is the mining method used at Mattabi. In the Upper Mine (above the 900 level) stopes are 76 m high.

Tonnage

The Mattabi mine produces 36,300 tonnes of ore per month.

Surface Facilities

The surface facilities at Mattabi include a mill, currently producing lead and zinc concentrates, a plant services building, and various shops and offices. Figure 6 illustrates the layout of surface facilities at Mattabi.

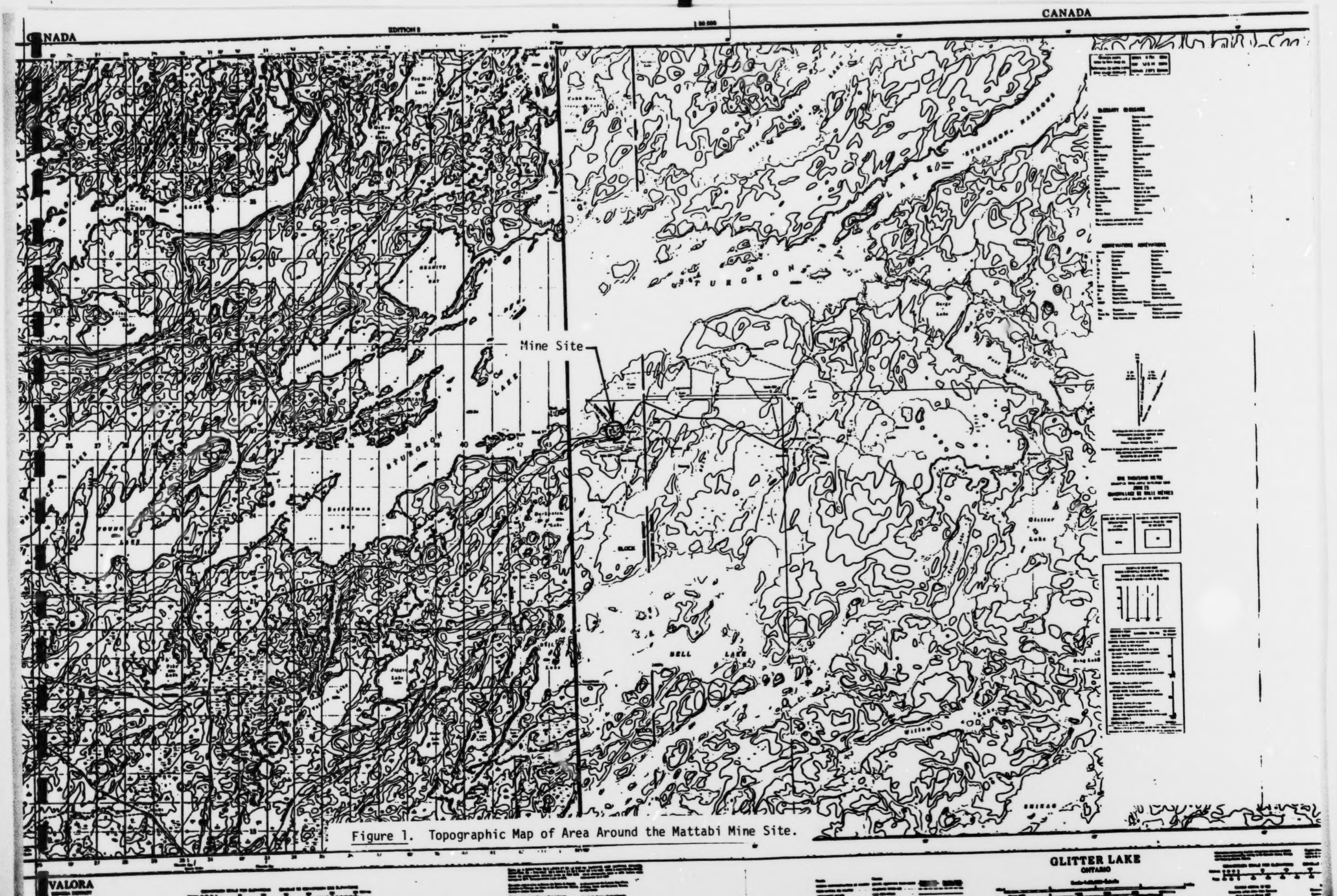

CANADA
CANADA
EDITION 2
1:50 000
Mine Site
STURGEON
LAKE
STURGEON
STURGEON
BELL LAKE
Glitter Lake
GLITTER LAKE
ONTARIO
VALORA

Figure 1. Topographic Map of Area Around the Mattabi Mine Site.

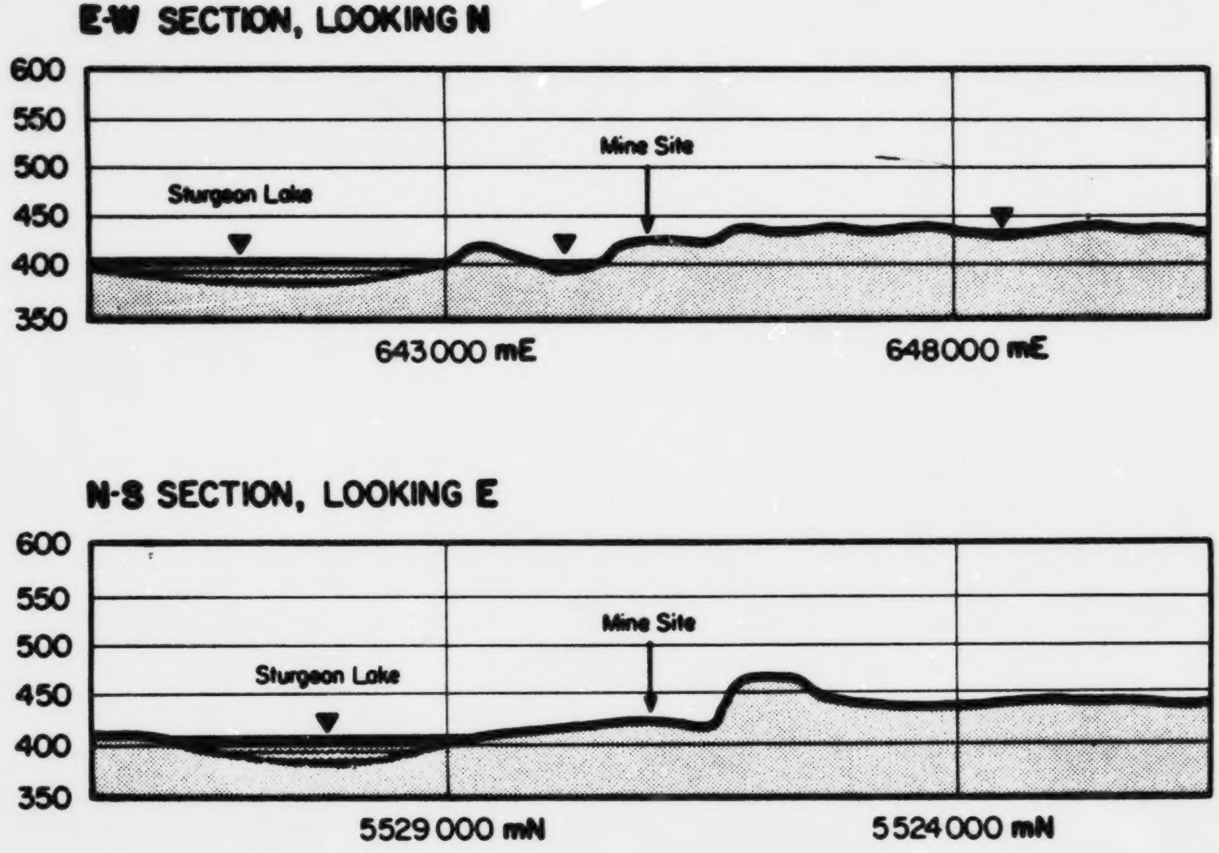
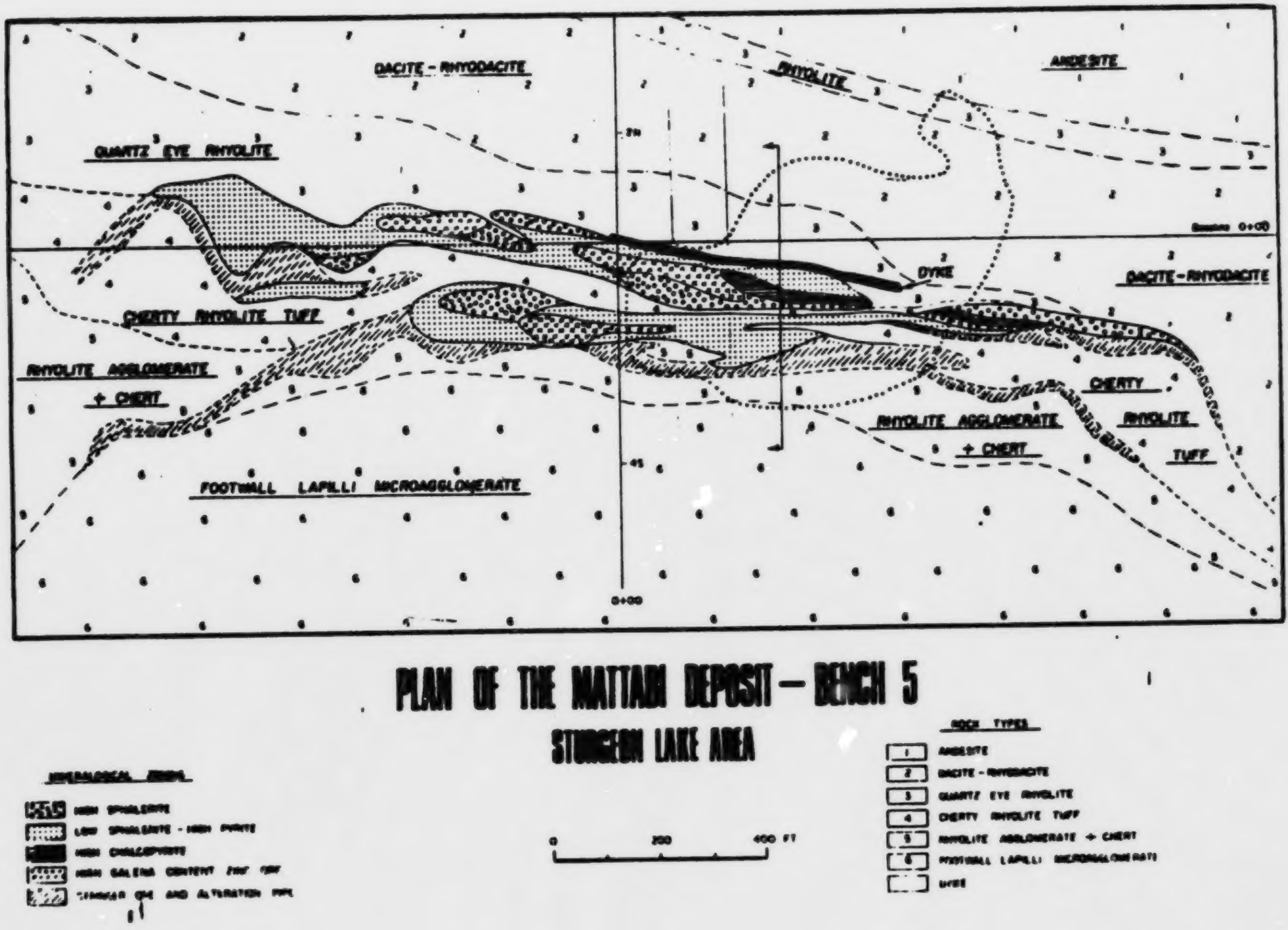


Figure 2. Topographic Sections Through the Mattabi Mine Site.



PLAN OF THE MATTABI DEPOSIT — BENCH 5

STURGEON LAKE AREA

DACITE - RHYODACITE
QUARTZ EYE RHYOLITE
RHYOLITE
ANDESITE
DYKE
DACITE - RHYODACITE
CHERTY RHYOLITE TUFF
RHYOLITE AGGLOMERATE
+ CHERT
RHYOLITE AGGLOMERATE
+ CHERT
CHERTY
RHYOLITE
TUFF
FOOTWALL LAPILLI MICROAGGLOMERATE
Section 0+00
0+00
MINERALOGICAL ZONES
HIGH SPHALERITE
LOW SPHALERITE - HIGH PYRITE
HIGH CHALCOPYRITE
HIGH GALENA CONTENT
STRINGER ORE AND ALTERATION PIPE
0
200
400 FT
ROCK TYPES
1 ANDESITE
2 DACITE - RHYODACITE
3 QUARTZ EYE RHYOLITE
4 CHERTY RHYOLITE TUFF
5 RHYOLITE AGGLOMERATE + CHERT
6 FOOTWALL LAPILLI MICROAGGLOMERATE
DYKE

Figure 3. Geological Plan of the Mattabi Deposit.

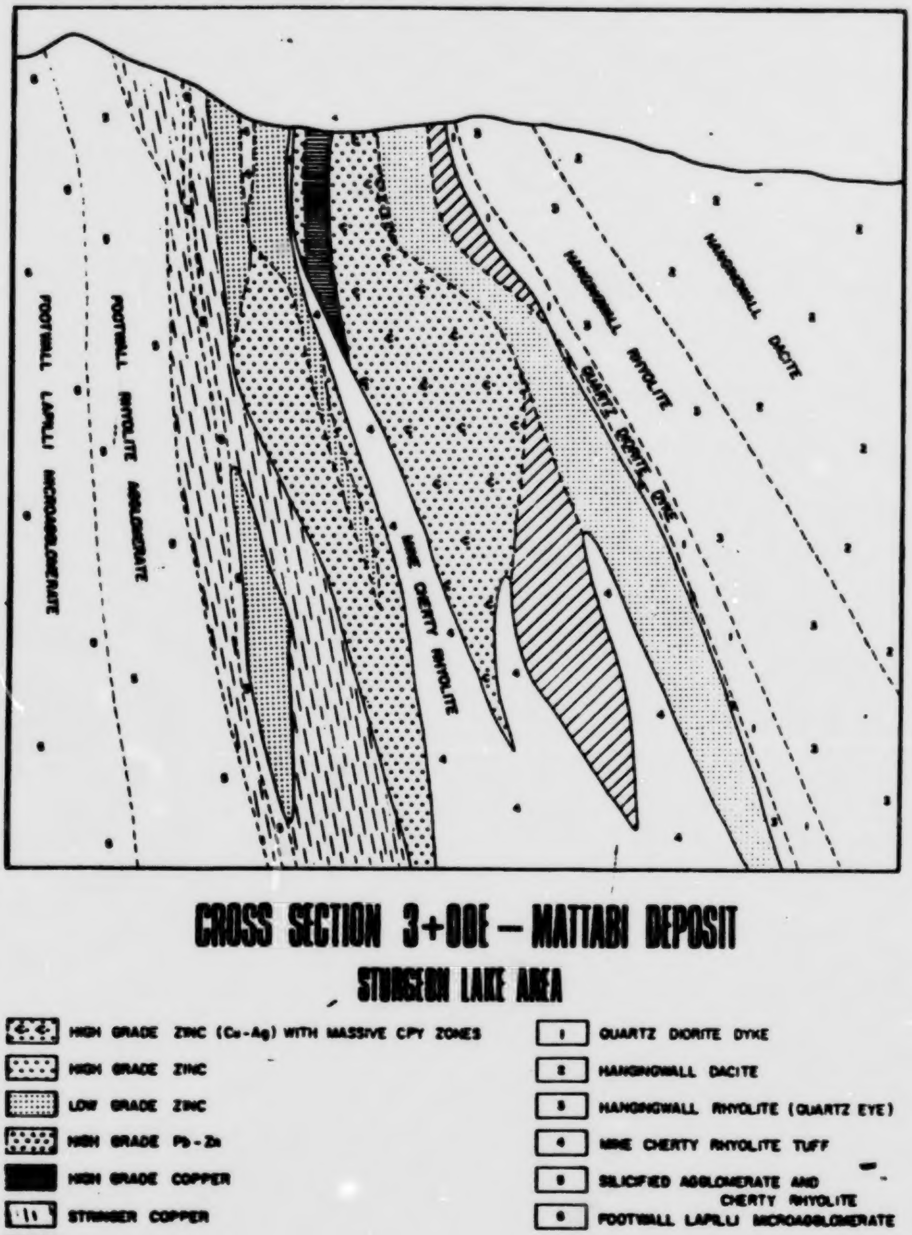

CROSS SECTION 3+00E – MATTABI DEPOSIT
STURGEON LAKE AREA
FOOTWALL LAPILLI MICROAGGLOMERATE
FOOTWALL RHYOLITE AGGLOMERATE
MINE CHERTY RHYOLITE
HANGINGWALL RHYOLITE
QUARTZ DIORITE DYKE
HANGINGWALL DACITE
HIGH GRADE ZINC (Cu-Ag) WITH MASSIVE CPY ZONES
HIGH GRADE ZINC
LOW GRADE ZINC
HIGH GRADE Pb-Zn
HIGH GRADE COPPER
STRINGER COPPER
MASSIVE PYRITE
1 QUARTZ DIORITE DYKE
2 HANGINGWALL DACITE
3 HANGINGWALL RHYOLITE (QUARTZ EYE)
4 MINE CHERTY RHYOLITE TUFF
5 SILICIFIED AGGLOMERATE AND CHERTY RHYOLITE
6 FOOTWALL LAPILLI MICROAGGLOMERATE
0
100
200 FT

Figure 4. Geological Section of the Mattabi Deposit.

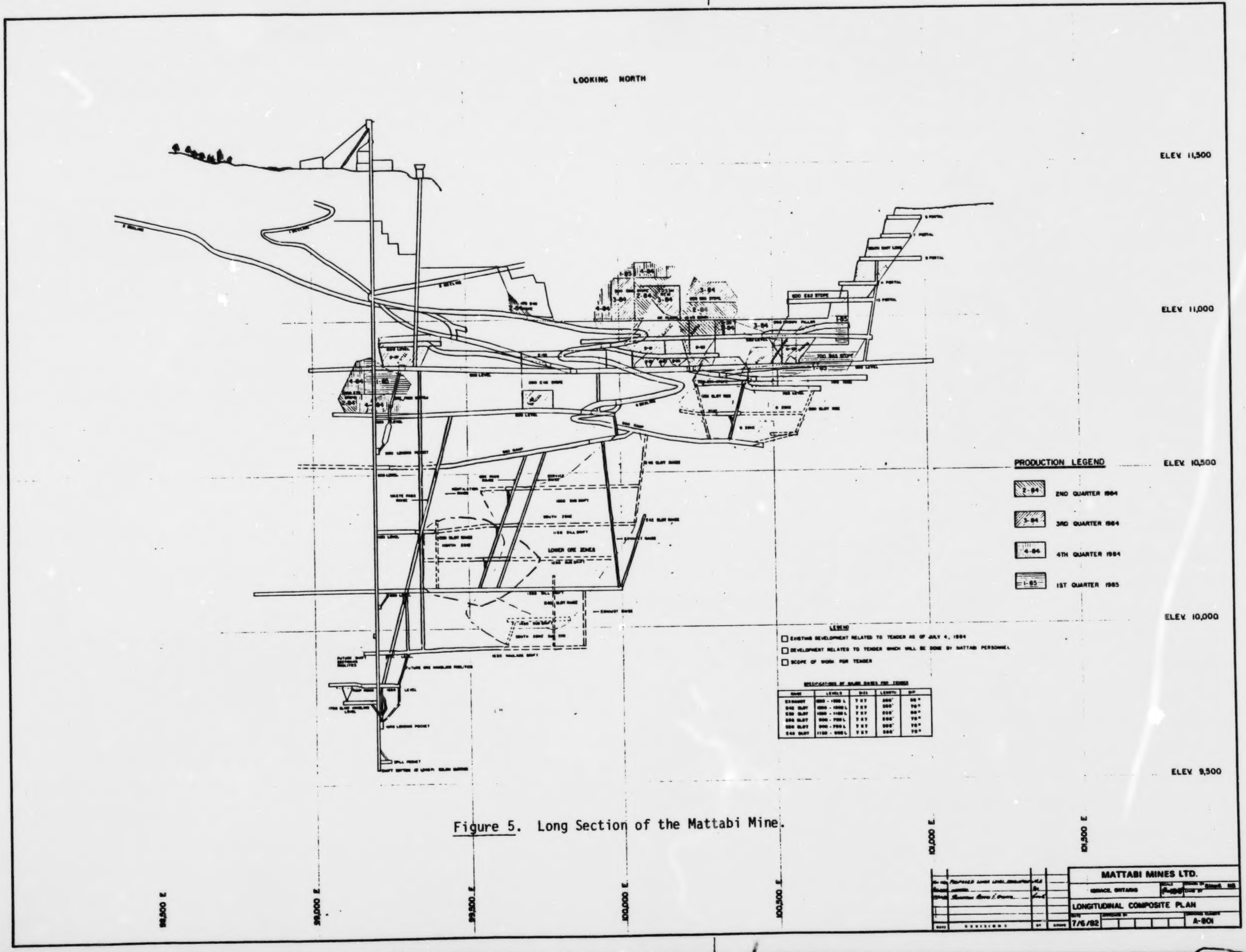

LOOKING NORTH
ELEV. 11,500
ELEV. 11,000
ELEV. 10,500
ELEV. 10,000
ELEV. 9,500
PRODUCTION LEGEND
2-84 2ND QUARTER 1984
3-84 3RD QUARTER 1984
4-84 4TH QUARTER 1984
1-85 1ST QUARTER 1985
LEGEND
☐ EXISTING DEVELOPMENT RELATED TO TENDER AS OF JULY 4, 1984
☐ DEVELOPMENT RELATED TO TENDER WHICH WILL BE DONE BY MATTABI PERSONNEL
☐ SCOPE OF WORK FOR TENDER
LOWER ORE ZONES
98,500 E
99,000 E
99,500 E
100,000 E
100,500 E
101,000 E
101,500 E
MATTABI MINES LTD.
IGNACE, ONTARIO
LONGITUDINAL COMPOSITE PLAN
7/6/82
A-801

Figure 5. Long Section of the Mattabi Mine.

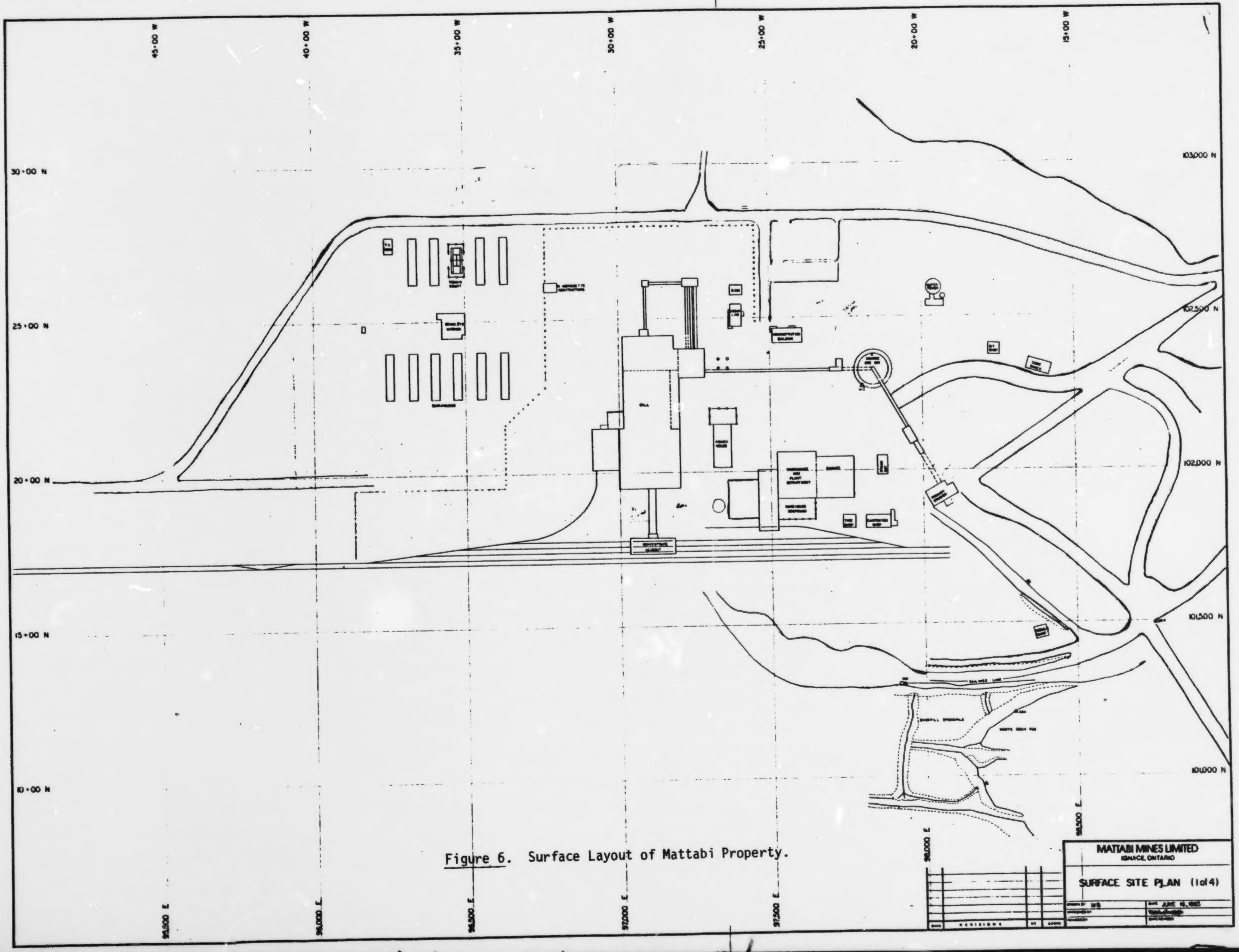

45+00 W
40+00 W
35+00 W
30+00 W
25+00 W
20+00 W
15+00 W
30+00 N
25+00 N
20+00 N
15+00 N
10+00 N
103,000 N
102,500 N
102,000 N
101,500 N
101,000 N
95,500 E
96,000 E
96,500 E
97,000 E
97,500 E
98,000 E
98,500 E
MILL
MATTABI MINES LIMITED
IGNACE, ONTARIO
SURFACE SITE PLAN (1of4)
REVISIONS
MB
JUNE 15, 1983

Figure 6. Surface Layout of Mattabi Property.

GEOLOGY

Mineralogy

High Sphalerite Ore : This ore is typically composed of 50 to 75 pct sphalerite, 20 to 45 pct pyrite, 2 to 5 pct gangue and accessory to 5 pct chalcopyrite.

Pyritic Ore : Massive pyrite ore zones contain fine to coarse pyrite with up to 10 pct sphalerite forming a matrix. Some areas contain up to 50 pct pyrrhotite.

Massive Chalcopyrite : This ore contains 75 to 90 pct chalcopyrite with streaks of sphalerite and accessory arsenopyrite.

Galena Ore : Ore from these zones contains 25 pct galena, 50 pct sphalerite, 10 pct pyrite, 1 to 2 pct tetrahedrite-tennantite and 2 to 3 pct arsenopyrite. The tetrahedrite-tennantite contains rare bournonite inclusions, and accessory boulongerite is also reported in this zone.

Chalcopyrite Stringer Ore : This ore is composed of course grained chalcopyrite with minor pyrite, pyrrhotite and accessory arsenopyrite. Magnetite is present in a few veins.

Gangue : Quartz is the most abundant gangue mineral, followed by chlorite, sericite, garnet, amphibole, biotite, anthophyllite and gahnite (a zinc spinel).

Alteration Pipe : Alteration is confined to the rocks underlying the Mattabi deposit. Sericite is the most common alteration mineral (up to 50 pct) followed by carbonate (close to 50 pct of some samples) choritoid (up to 15 pct), chlorite, kyanite, andalusite and almandine.

Structure

The mine area lacks structural complexity. Faults of major displacement are absent, and folding is on a very large scale. Minor folds appear related to emplacement of intrusive rocks. In general, the mine has competent ground, and no ground control problems have been experienced.

Ambient ground temperature is about 10°C.

MINE WATER

An average of 545,000 l/day of water is pumped from the Mattabi mine. The water underground has a pH of between 3.2 and 3.4 and it is treated with lime before it is pumped to the surface.

VENTILATION

71 m^3/s of fresh air is supplied to the mine. 7 to 9.5 m^3/s is exhausted up the shaft and the remainder to the open pit.

POTENTIAL ZONES FOR BACTERIAL LEACHING

375,000 tonnes of Upper Mine proven sub-ore material grading 4.5% Zn, 0.94% Cu, 0.49% Pb and 85.7% g/t Ag represents potential ore for leaching tests. In particular, the E zone above the 950 level contains about 182,000 tonnes grading 1 to 1.5% Cu and 2% Zn. This zone is accessible from existing mine development. Figure 7 shows the location of the E zone on a sample geological section of the mine.

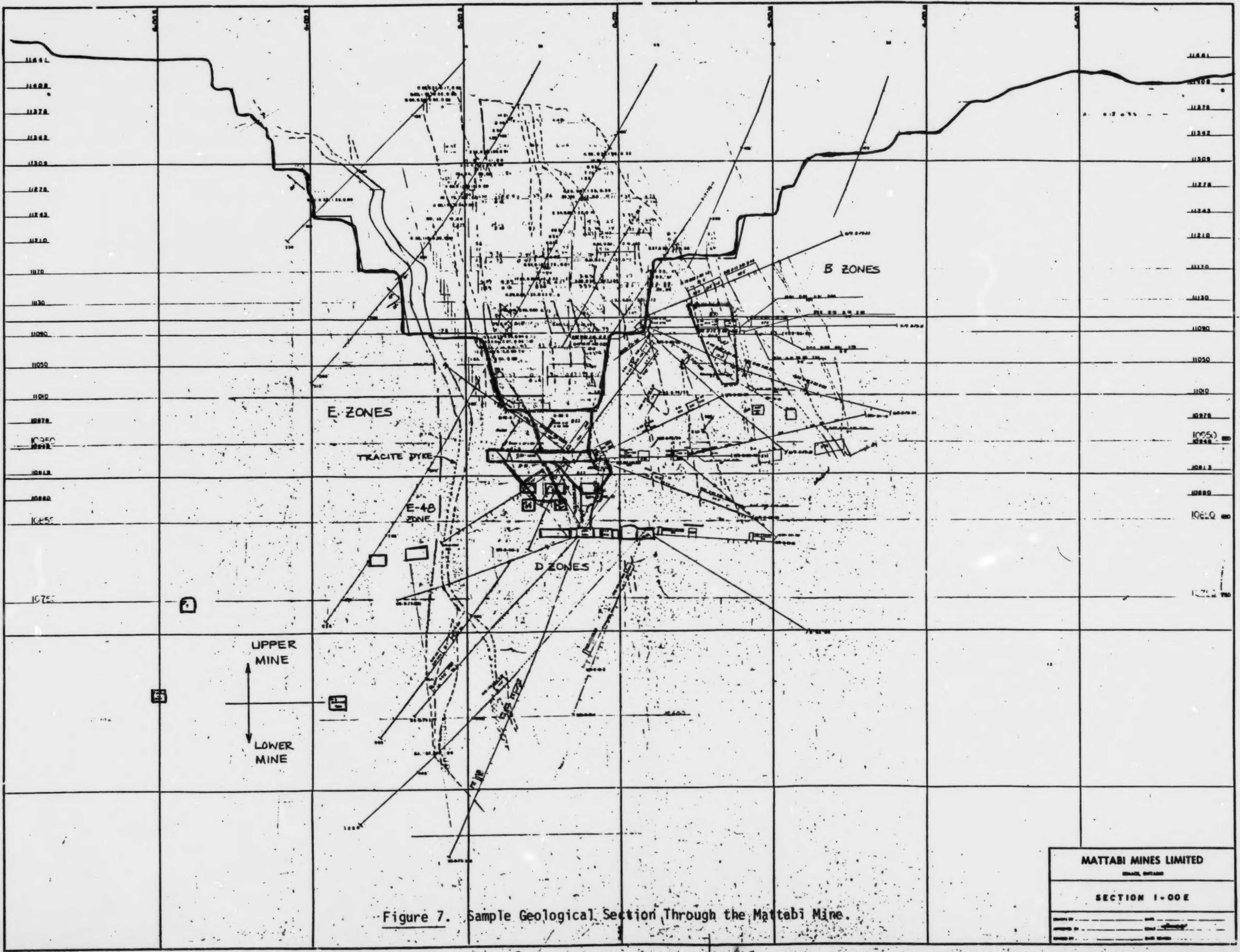

B ZONES
E ZONES
TRACITE DYKE
E-48 ZONE
D ZONES
UPPER MINE
LOWER MINE
MATTABI MINES LIMITED
SECTION 1-00E

Figure 7. Sample Geological Section Through the Mattabi Mine.

APPENDIX G

MINE SITE REPORT FOR
THE COPPER MINE
(PAMOUR PORCUPINE MINES)

PAMOUR PORCUPINE MINES LIMITED

COPPER MINE

MINE SITE

Location

The Copper Mine is a copper ore zone on the property of Schumacher Division Mine in Schumacher, near Timmins.

Topography

Relatively gentle topography surrounds the Timmins area, as shown in Figure 1. Figure 2 contains north-south and east-west sections through the mine site.

Climatology

Temperature and precipitation data are not readily available for the Copper Mine area.

MINE CONFIGURATION

Orebody

The Copper Mine orebody is contained within a mineral assemblage known as the Pearl Lake Porphyry. It has a strike length (east-west) of 396 m, is about 1070 m high (1125 level to 4625 level) and varies in width. The orebody dips near vertical and plunges at 60° to the east.

Mining Method

Small diameter longhole was the main mining method used while the Copper Mine was in production. Some large diameter (ITH, or in-the-hole) stoping was also practiced.

Tonnage

Mining rate during production at the Copper Mine varied between 272,000 and 698,000 tonnes per year. The Copper Mine was shut down in March 1982.

GEOLOGY

Mineralogy

Figures 3 and 4 illustrate the geology of the Pearl Lake Porphyry.

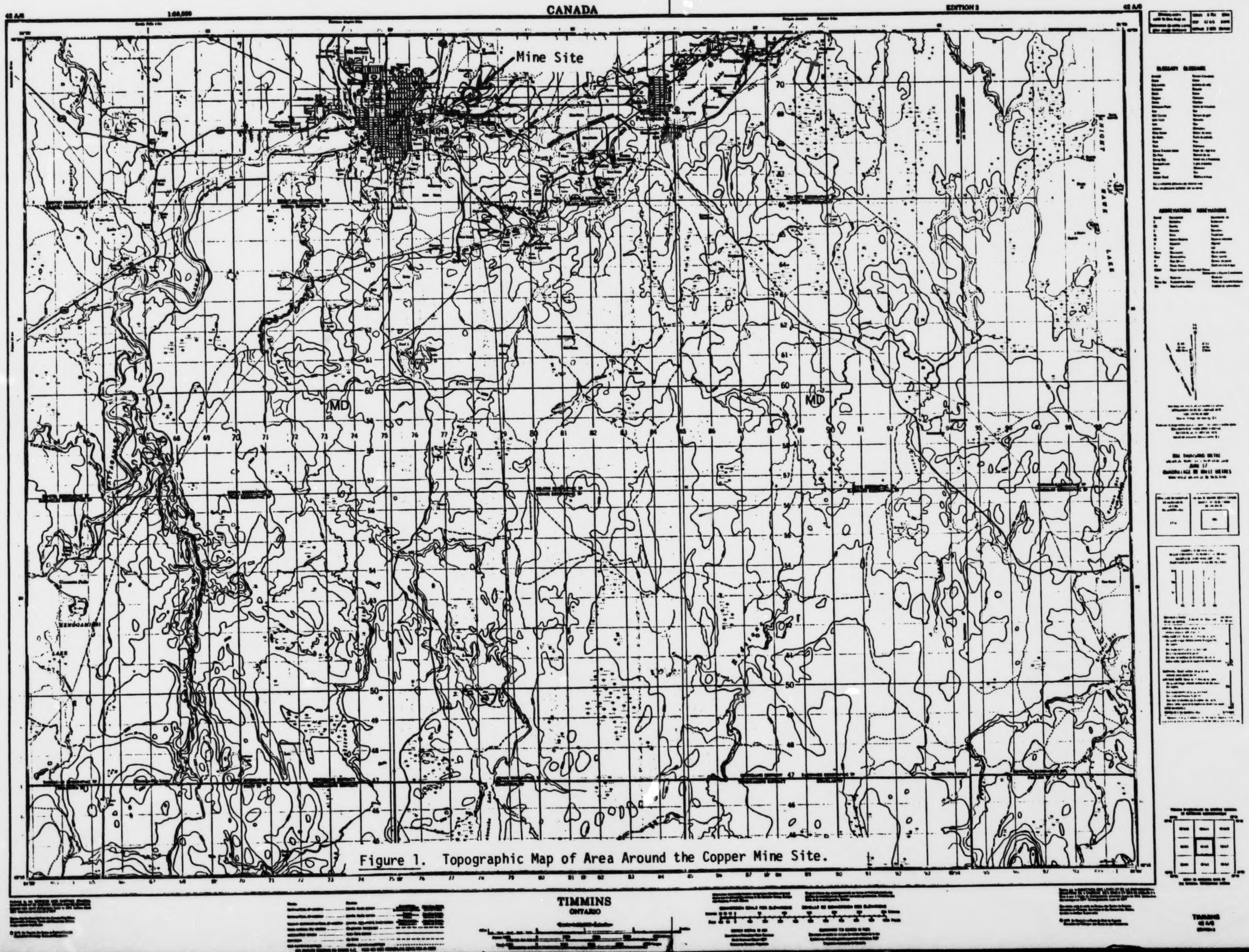
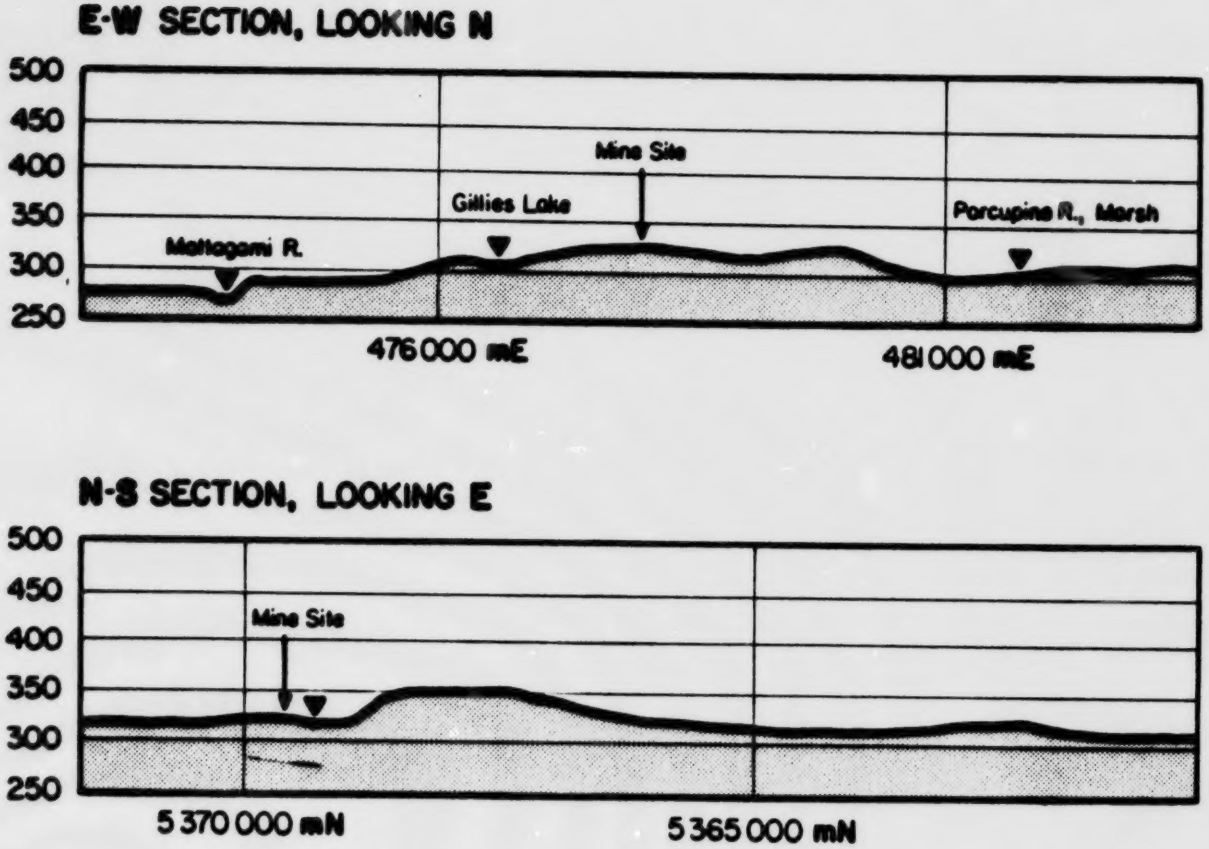

CANADA
EDITION 3
Mine Site
TIMMINS
MD
MD
TIMMINS
ONTARIO

Figure 1. Topographic Map of Area Around the Copper Mine Site.


E-W SECTION, LOOKING N

N-S SECTION, LOOKING E

Figure 2. Topographic Sections Through the Copper Mine Site.

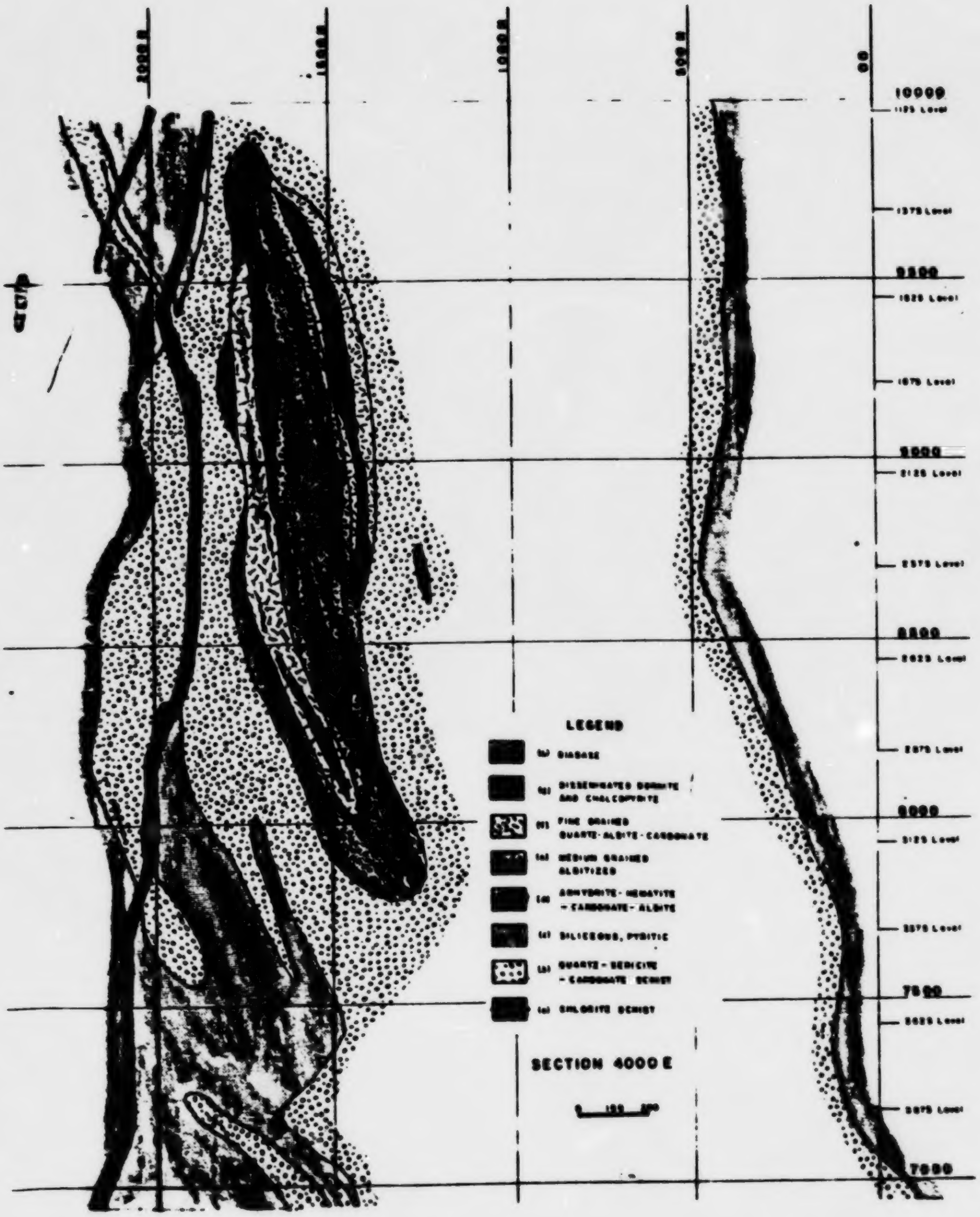

2000 S
1500 S
1000 S
500 S
00
10000
1125 Level
1375 Level
9500
1625 Level
1875 Level
9000
2125 Level
2375 Level
8500
2625 Level
2875 Level
8000
3125 Level
3375 Level
7500
3625 Level
3875 Level
7000
LEGEND
DIABASE
DISSEMINATED BORNITE AND CHALCOPYRITE
FINE GRAINED QUARTZ-ALBITE-CARBONATE
MEDIUM GRAINED ALBITIZED
ANHYDRITE-HEMATITE-CARBONATE-ALBITE
SILICEOUS, PYRITIC
QUARTZ-SERICITE-CARBONATE SCHIST
CHLORITE SCHIST
SECTION 4000 E
0 100 200

Figure 3. Geological Section of the Copper Ore Zone.

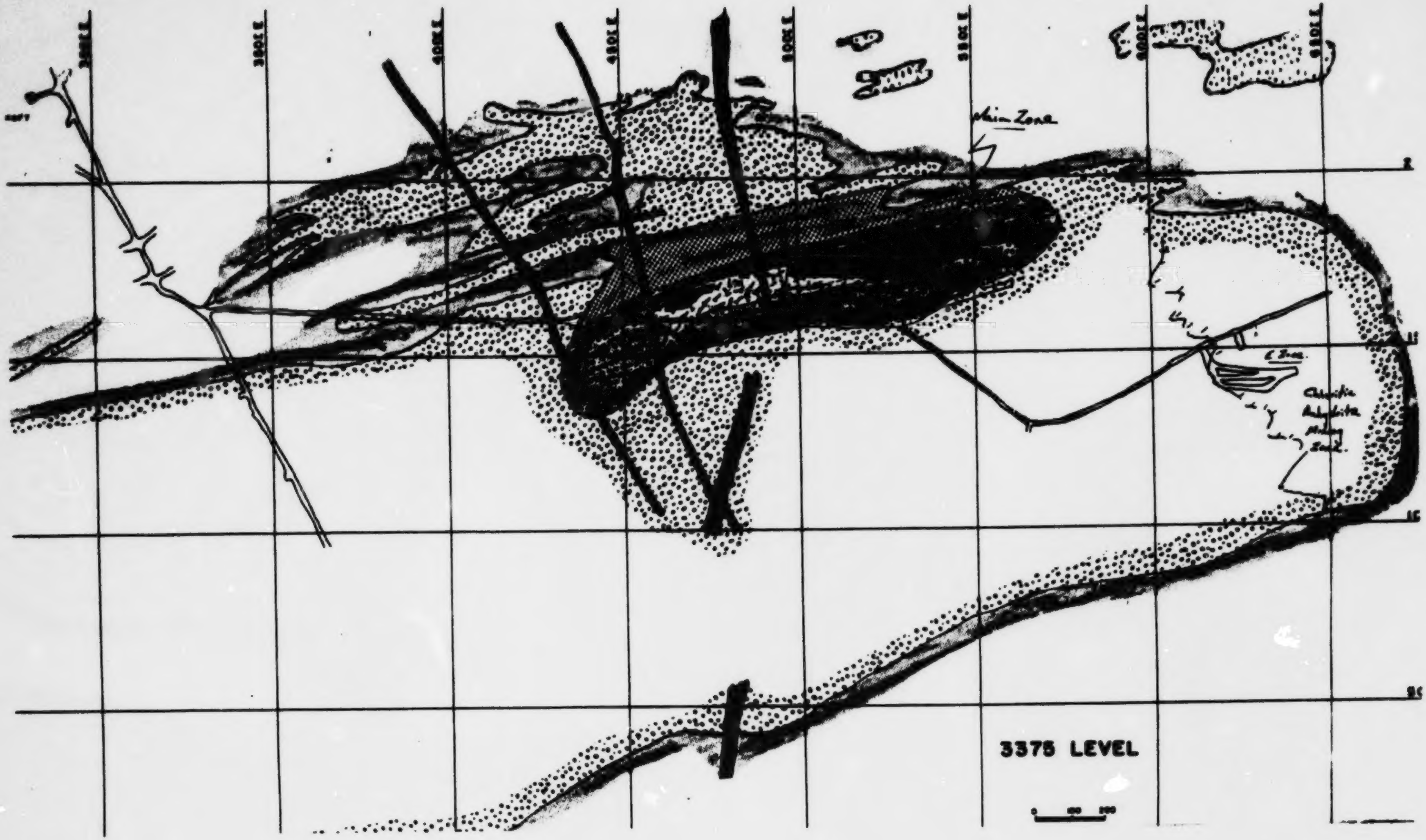

Main Zone
C Zone.
Chlorite
Anhydrite
Mining
Zone.
3375 LEVEL
0 100 200

Figure 4. Geological Plan of the Copper Ore Zone.

Ore Minerals : Major ore minerals are chalcopyrite, pyrite, molybdenite, bornite and tetrahedrite. Secondary minerals are albite, quartz, carbonate and anhydrite.

Gangue : The albitic zones which surround the ore contain albite, quartz, carbonate, anhydrite and some hematite. The bulk of the Pearl Lake Porphyry is composed of quartz-sericite schist which also contains quartz eyes, carbonate and anhydrite.

Structure

Two major periods of folding in the area have produced two prominent sets of shearing. One set strikes N75°W and dips 67° northeast, and the second strikes N65°E and dips at 73° to the southeast. There is a third set of minor shear planes striking east with a vertical dip.

During mining, sloughing from the walls created high (20 to 25 pct) dilution in stopes. The ground in the deeper eastern portion of the mine is fractured and heavily foliated along strike, although there are no major faults in this area.

POTENTIAL ZONES FOR BACTERIAL LEACHING

The total mineral inventory (proven, probable and possible ore) for the Copper Mine is 1,236,000 tonnes grading 0.47% Cu and 1.37 g/t Au. These figures include 84,000 tonnes of broken ore remaining in stopes. Most of this material is in the deeper eastern part of the mine.

One potential test area is the 3382 stope (see the mine long section in Figure 5). This stope contains 149,000 tonnes in place and 20,750 tonnes of broken ore, totalling 169,750 tonnes grading 0.45% Cu and 0.55 g/t Au. Figure 6 is a representative geological section through this stope.

Ambient air temperature in this area is 13°Cto 16°C, and 4.7 m^3/s of air currently ventilates the 3382 stope.

The poor ground conditions referred to earlier would create the possibly of migration of leaching solutions.

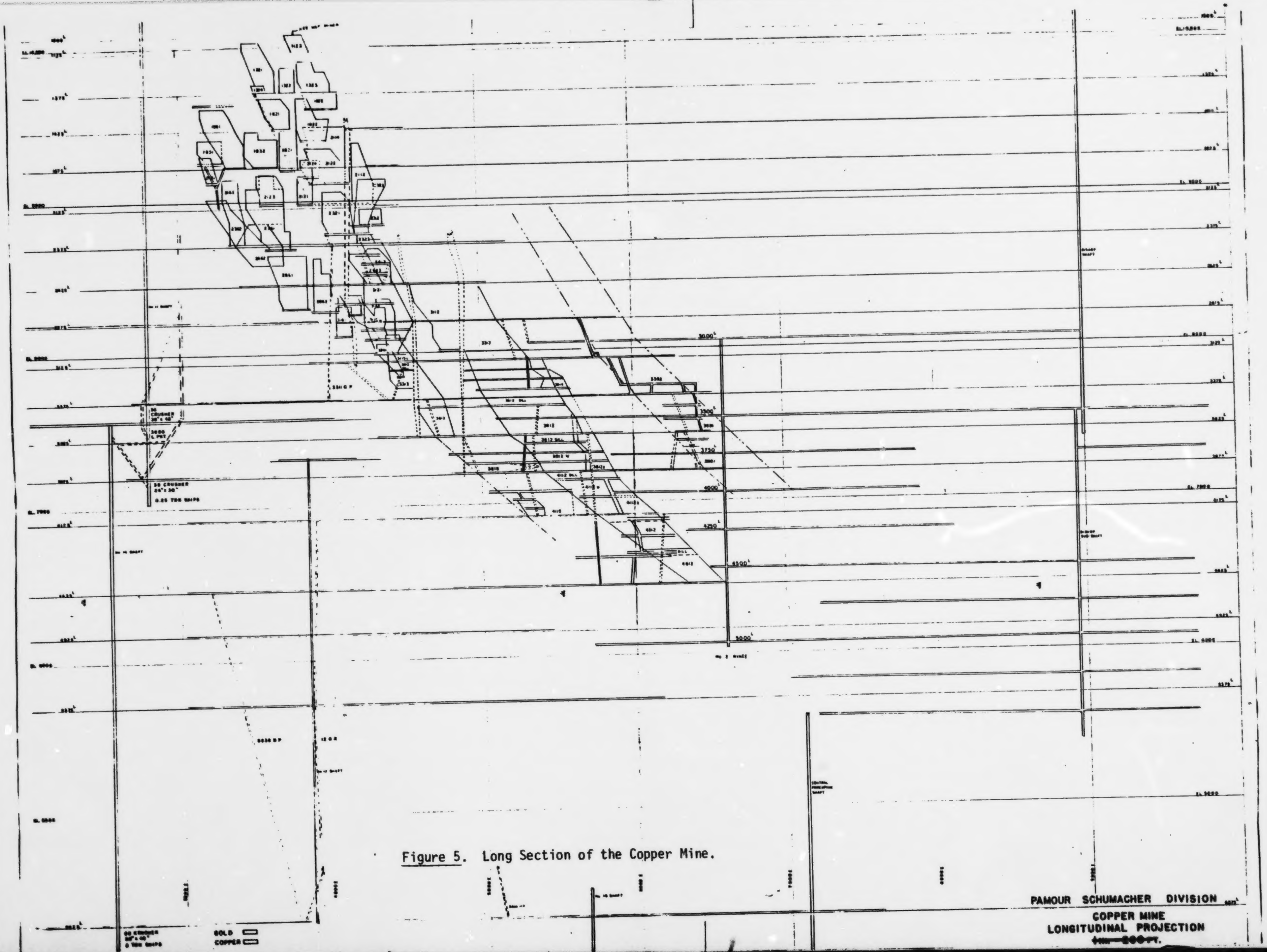

3000
3500
3750
4000
4250
4500
5000
3312
3612
CRUSHER
GOLD
COPPER
PAMOUR SCHUMACHER DIVISION
COPPER MINE
LONGITUDINAL PROJECTION

Figure 5. Long Section of the Copper Mine.

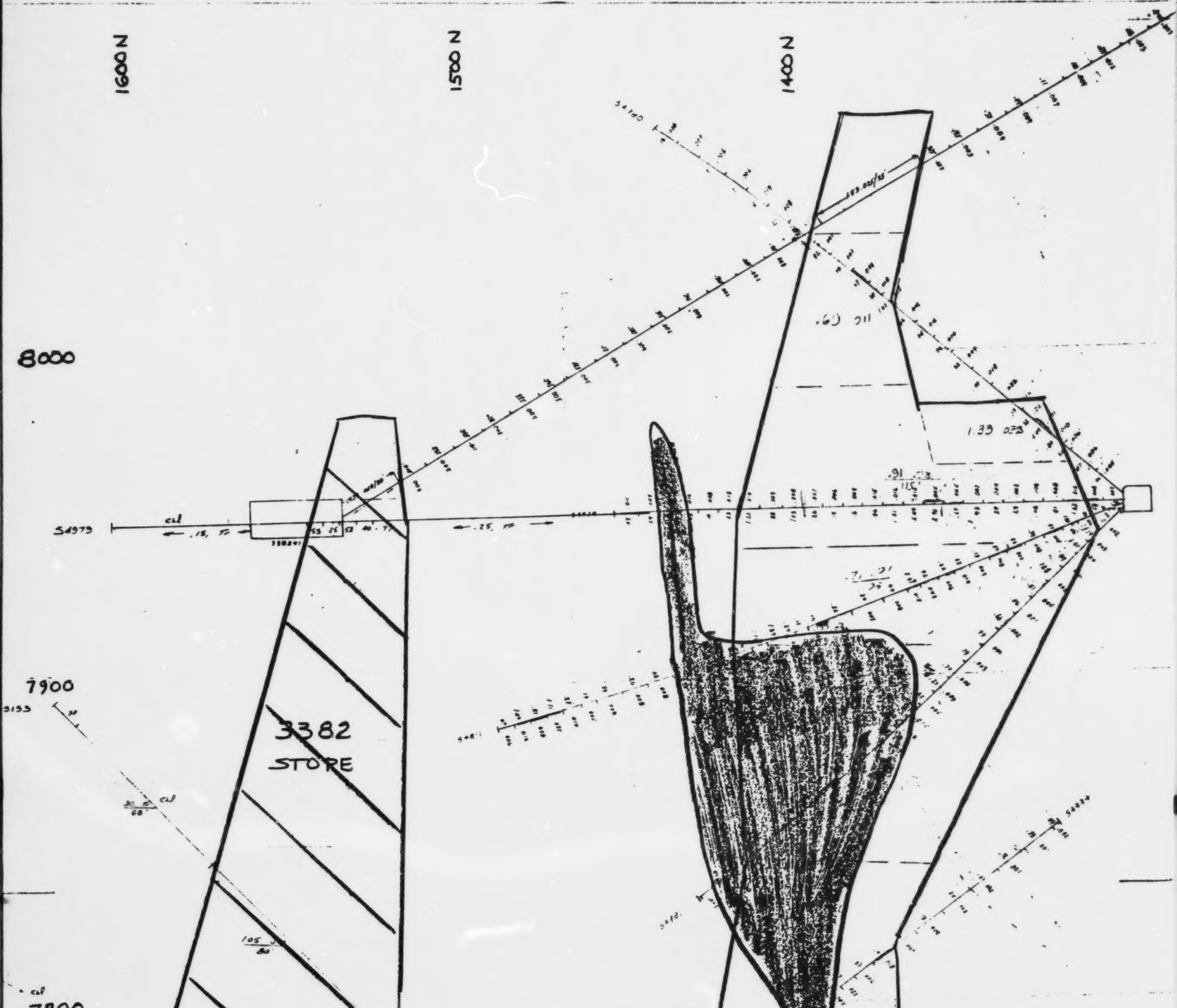

1600 N
1500 N
1400 N
8000
7900
7800
7700
3382
STOPE
6200 E
Grey Porphyry
Alb-Qtz
Pink Porphyry
Alb-An-Hem.
chlorite Schist
Basalt Lava
S4979
S4173
S4903
1.33 023
.63 211

Figure 6. Sample Geological Section of 3382 Stope.

APPENDIX H

LIST OF MINERALS REFERRED TO IN THIS REPORT, WITH THEIR FORMULAE.

MINERAL NAME	FORMULA
Albite	An_{0-10}
Allanite	$(Ca,Fe^{II})_2(Al,Ce,Fe^{III})_3(OH)(SiO_4)_3$
Almandite	$Fe_3Al_2(SiO_4)_3$
Andalusite	Al_2SiO_5
Anhydrite	$CaSO_4$
Anthophyllite	$(Mg,Fe)_7(OH)_2(Si_4O_{11})_2$
Arsenopyrite	$FeAsS$
Biotite	$K_2(Mg,Fe)_2(OH)_2(AlSi_3O_{10})$
Bornite	Cu_5FeS_4
Boulangerite	$Pb_5Sb_4S_{11}$
Bournonite	$PbCuSbS_3$
Chalcocite	Cu_2S
Chalcopyrite	$CuFeS_2$
Chlorite	$(Mg,Fe,Al)_6(Al,Si)_4O_{10}(OH)_8$
Chloritoid	$H_2(Fe,Mg,Mn)Al_2SiO_7$
Clinozoisite	$Ca_2Al_3(OH)(SiO_4)_3$
Covellite	CuS
Diopside	$Ca(Mg,Fe)\ (SiO_3)_2$
Epidote	$Ca_2(Al,Fe)_3(OH)(SiO_4)_3$
Gahnite	$ZnAl_2O_4$
Galena	PbS
Garnet(Spessartine)	$Mn_3Al_2(SiO_4)_3$
Grossularite	$Ca_3Al_2(SiO_4)_3$
Grunerite	$Fe_7(OH)_2(Si_4O_{11})_2$

Hematite	Fe_2O_3
Kaolinite	$Al_2O_3.2SiO_2.2H_2O$
Kyanite	Al_2SiO_5
Magnetite	Fe_3O_4
Marcasite	FeS_2
Molybdenite	MoS_2
Pyrargyrite	Ag_3SbS_3
Pyrite	FeS_2
Pyrrhotite	$Fe_{(1 - x)}S$
Quartz	SiO_2
Scheelite	$CaWO_4$
Sericite	$KAl_2(OH)_2(AlSi_3O_{10})$
Sphalerite	$(Zn,Fe)S$
Tennantite	$Cu_{12}As_4S_{13}$
Tetrahedrite	$Cu_{12}Sb_4S_{13}$
Thompsonite	$NaCa_2Al_5(SiO_4)_56H_2O$
Wollastonite	$CaSiO_3$

Volume

2

noranda

CONTRACT REPORT NO. 0SQ84-00231

CONTRACT RESEARCH REPORT
N-8515 C.R. #85-02.

A PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT

DSS FILE NO. 07SQ. 23440-4-9071

MILESTONE 2 REPORT

PRE-FEASIBILITY OF UNDERGROUND OPERATIONS

DECLASSIFIED
DATE 8/2/86
AUTHORIZED BY R. McCready

Project N-8515

March 1985

MICROMEDIA

Mining Technology Division
Centre de Recherche Noranda
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H9R 1G5

62pp.

CONFIDENTIAL

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SUMMARY

During Phase I of this project, the Geco mine in Manitouwadge, Ontario was selected as the most favourable Canadian Noranda operation to conduct a pre-feasibility study of bacterially-assisted underground leaching of a low grade sulphide copper deposit. The purpose of this report, Phase II of the investigation, is to estimate the costs to prepare stopes for leaching.

A representative block in the sericite schist zone containing low grade sulphide copper (<1%) was selected for this phase of the study. This block is located between level 2650 and 2850 and between section 64+00 and 68+00. Development plans containing access drifts at the top and at the bottom of the stopes, slots for blasting and drilling and blasting plans were established for three stopes. In these stopes, 94300 tonnes of ore are available for leaching. The estimated costs to prepare these stopes are $1,390,000 which is $14.74/tonne. These costs also include the excavation of a sump for the leachate and the construction of bulkheads in the access drift at the bottom of the stopes to automatically stop any leachate spill in case of pump stopage.

According to the permeability testing done by Geomine, no significant leachate loss are expected through the rock mass.

SOMMAIRE

Dans la phase I de ce projet, la mine Geco de Manitouwadge en Ontario, a été choisie comme étant la mine canadienne du groupe Noranda la plus propice pour effectuer une étude de pré-faisabilité de lixiviation souterraine bactériologiquement assistée sur un dépôt du sulfure de cuivre à faible teneur. La présente phase II du projet a pour but d'estimer les coûts de préparation de chantiers pour la lixiviation.

Un bloc représentatif de la zone de schiste à séricite contenant des sulfures de cuivre à faible teneur (<1%) a été choisi pour cette phase de l'étude. Ce bloc est situé entre les niveaux 2650 et 2850 et entre les sections 64+00 et 68+00. Les plans de développement contenant les galeries d'accès au haut et au bas des chantiers, les ouvertures pour le sautage et les plans de forage et de sautage ont été faits pour trois chantiers. Dans ces chantiers, 94300 tonnes de minerai sont disponibles pour la lixiviation. Les coûts de préparation de ces chantiers sont estimés à 1390000 $, soit 14.74 $ par tonne. Ces coûts incluent aussi le creusage d'un bassin collecteur pour les solutions et la construction de cloisons dans les galeries d'accès au bas des chantiers pour arrêter automatiquement l'écoulement des solutions en cas d'arrêt des pompes.

D'après les essais de perméabilité effectués par Géomines, aucune perte significative de solutions à travers la masse rocheuse n'est anticipée.

PART I

a) Presently, the project is behind schedule. The reason for the delay was explained during our meeting on December 17, 1984 and was stated in the Milestone I report.

b) The project is within budget.

c) The project is free of any areas of concern on which Crown assistance or guidance is required.

Milestone 3 report will follow shortly and Milestone 4 report is still planned for March 31, 1985.

PART II

a) Task 2, the Pre-Feasibility of Underground Operations, has been completed as required. Task 3 was conducted simultaneously and will soon be completed. Work will begin immediately on task 4. The whole project will be completed by the end of March, 1985.

b) No variation from the plan of work was made during the completion of task 2.

c) No major items of equipment were purchased during the completion of task 2.

d) Two trips were made to Geco Mine in relation with this phase. One was by our personnel to gather all necessary information to layout stopes and to estimate costs for the underground operation. The other was made by our hydrology consultant to evaluate the permeability of the leaching area.

PART III

A copy of the "Plan of Work and Cost" follows this page. It illustrates the present time allowance for each task and the work completed as of March 8, 1985.

TABLE A

Supply and Services Canada Approvisionnements et Services Canada

TIME/LOST CHART - *

MAS 9143 3 (4/81)

CONTRACT PLAN AND REPORT FORM – FORMULE DE PROJET DE CONTRAT ET DE RAPPORT

CONTRACT TITLE – TITRE DU CONTRAT 'PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT'

CONTRACTOR – ENTREPRENEUR Centre de Recherche Noranda-NORANDA INC.

FILE NO. - Nº DU DOSSIER: 07SQ.23110-4-9071

SERIAL NO. - Nº DE SÉRIE:

TASK DESCRIPTION DESCRIPTION DE LA TÂCHE	1984 1. July	2 Aug.	3 Sept.	4 Oct.	5 Nov.	6 Dec.	1985 7 Jan.	8 Feb.	9 Mar.	10 Apr	11 May	12	TASK COST – COÛT DE LA TÂCHE PLANNED PRÉVU À L'ORIGINE	ACTUAL RÉELLEMENT ENGAGÉ	E.T.C. À VENIR
1. Mine Site Selection			▬	▬	▬										
2. Pre-feasibility of Underground Operations			- - -	- - -	- - - ▬	▬	▬	▬	▬						
3. Pre-feasibility of Met. Circuit			- - -	- - -	- - - ▬	▬	▬	▬	▬						
4. Sensitivity Analysis							- - -	- - -	- - - ■ - - -	- - -					
PLANNED COST BY MONTH COÛT PRÉVU POUR CHAQUE MOIS			$20,000	$20,000	$15,000	$15,000	$15,000	$10.000	$ 5,000				PLAN	▨	▨
REV. PRÉVISION. E.T.C. BY MONTH PRÉVISION POUR CH. MOIS													▨	ACTUAL COÛTS RÉELS	E.T.C. À VENIR

J. Martel

PART IV: PHASE REPORT

1. INTRODUCTION

The purpose of this study is to evaluate the feasibility of recovering copper, which otherwise would not be recovered in an operating mine, using bacterially assisted leaching, from a low grade disseminated sulphide zone. The production generated from this process would represent additional revenue and would not require an increase in fixed costs such as administration and plant overhead costs.

This part of the study, "Pre-Feasibility of Underground Operations", has the purpose of estimating basic mine costs for the sensitivity analysis to be done in Phase IV.

From Phase I of this project, the Geco mine in Manitouwadge, Ontario was selected as the most favourable Noranda mine for this investigation.

2. GEOLOGY

R.G. Friesen, G.A. Pierce and R.M. Weeks have well described the Geco geology in a paper entitled "The Geology of the Geco Base Metal Deposit" (1982).

The regional geological structure consists of a broad easterly plunging synform. The Geco orebody follows a dragfold in the south limb of this synform. It is one vertical, lenticular, continuous zone of mineralization which is interrupted by several diabase dykes and is offset by the Fox Creek Fault. The average horizontal length on any level is approximately 730 m and the average width is about 20 m. The bottom of the orebody plunges to the east at an average of 35° as it follows the S-shaped dragfold which exists west of the orebody on each level.

The orebody is made up of a core of massive sulphides consisting of pyrite, pyrrhotite, sphalerite, chalcopyrite, galena and minor amounts of gold. Appreciable silver is present, associated with the chalcopyrite and galena. The remainder of the massive core is made up of wall rock inclusions. A zone of disseminated sulphides, containing no appreciable precious metal content, in sericite schist forms an envelope around the massive sulphide core and it is this zone that is investigated in this study.

3. MINING BACKGROUND

Since production started in 1957, some 36.2 million tonnes of ore have been mined from the massive sulphide ore body.

Bulk blasthole mining has accounted for 95 percent of the ore extracted at Geco to date. Primary stopes were 21 m wide, 91 m high transverse slices at 37 and 46 m intervals along strike. These have all been mined and filled with waste rock fill. Hydraulically placed 30:1 cemented mill tailings were used to stabilize the walls of these stopes, and the pillars between them are now being mined.

The remaining 5 percent of extraction is by cut and fill mining, which is used where the ore is too narrow for longitudinal blasthole stoping and more ground support is required. The reserves now stand at about 15.5 million tonnes.

4. STOPE LAYOUT

The area of concern in this study is a zone of disseminated sulphide in sericite schist which envelopes the massive sulphide core. Since this zone has not been of interest in the past, no reserve estimates had been made.

Since, at this stage, the economical size for a recovery plant is unknown, this study is conducted by laying out stopes in a limited area of the mine. The cost estimate arrived at for developing these stopes will be the basis of the sensitivity analysis to take place during Phase IV.

The area selected is a block bounded by sections 62+00 and 68+00 and by levels 2650 and 2850 containing about 1 x 10^6 tonnes grading about 0.5% Cu with another 1.5 x 10^6 tonnes of a similar grade located immediately above and below it. Access to this block is already available.

By arbitrarily selecting a cut-off grade of 0.5%, from experience at Geco, and their level intervals, three stopes are layed out 9.1 m in width, 31.4 m long and 61.0 m high. These grade 0.9%, 0.7% and 0.9% copper respectively, and are vertical. It is assumed that the solution will not seriously channel over the depth of the stopes.

5. DEVELOPMENT

The required development can be divided into level access, including sump excavation, and stope preparation. The proposed level access ensures continued access to 28-65 and 26-65 drifts on levels 2850 and 2650 respectively and therefore, will not hinder future drilling endeavours in this area. Sumps are to be excavated downstream from the leaching area.

Stope preparation includes:

- overcuts
- undercuts (draw points)
- Alimak raises
- sub-level development
- slots
- drilling and blasting, and
- bulkheads.

5.1 Level Access

Level development is required on both 2650 and 2850 levels. On 2650, approximately 140 m of 2.7 m x 2.7 m drifting is needed. Of this, 38 m is low grade ore containing about 810 tonnes of 0.4% copper and another 28 m containing 590 tonnes at 0.7%.

More extensive development is required on 2850 level. Here, 278 m representing about 5950 tonnes must be mined to yield access to the draw points which will become solution handling drifts once leaching begins. Of this projected development, 57% is in mineralization and the remaining 43% is in waste rock as detailed in Table 1.

A total of 418 m of drifting yielding approximately 4800 tonnes of ore at 0.6% Cu, and 4200 tonnes of waste is needed to access the planned stopes. Figures 1 and 2 indicate which development takes place in the mineralized zone.

5.2 Sump Excavation

A common sump having a holding capacity of more than 200 m^3 is planned to be excavated between 28-1 east drift and 28-61 crosscut. Any overflow in the pump area will drain down the raise at the end of 28-61 crosscut.

5.3 Stope Preparation

Schematics of fully prepared stopes are shown in Figures 3 and 4. Stopes 1 and 3 are developed in the same manner, whereas stope 2 required a slightly different approach because it tapers at the bottom, generally to the south, and is characterized by a block of low grade (0.1%) material at the bottom and in the centre. Consequently, two slots and a draw point at each end of the stope

TABLE 1

Summary of Required Level Access Development

LEVEL	IDENTIFICATION	LENGTH (m)	AMOUNT (tonnes)	GRADE (%)	TONNES OF COPPER
2650	26-1' XCUT	38.00	816.00	0.40	3.26
		74.00	1606.00	WASTE	0.00
	26-1' DR.	28.00	590.00	0.70	4.13
					0.00
2850	28-1' XCUT	32.00	688.00	0.40	2.75
		92.00	1966.00	WASTE	0.00
	28-2' XCUT	8.00	164.00	2.40	3.94
	28-3' XCUT	10.00	216.00	0.20	0.43
	28-4' XCUT	23.00	492.00	0.80	3.94
		15.00	328.00	WASTE	0.00
	28-5' XCUT*	n.a	n.a	n.a	0.00
	28-6' XCUT*	n.a	n.a	n.a	0.00
	28-1' DR.	43.00	936.00	0.80	7.49
		6.00	131.00	WASTE	0.00
	28-2' DR.	43.00	936.00	0.40	3.74
		6.00	131.00	WASTE	0.00
	TOTAL:	418.00	9000.00		29.68
		ORE:	4838.00		
			AVERAGE GRADE:		0.61

* included in 28-1' DR.

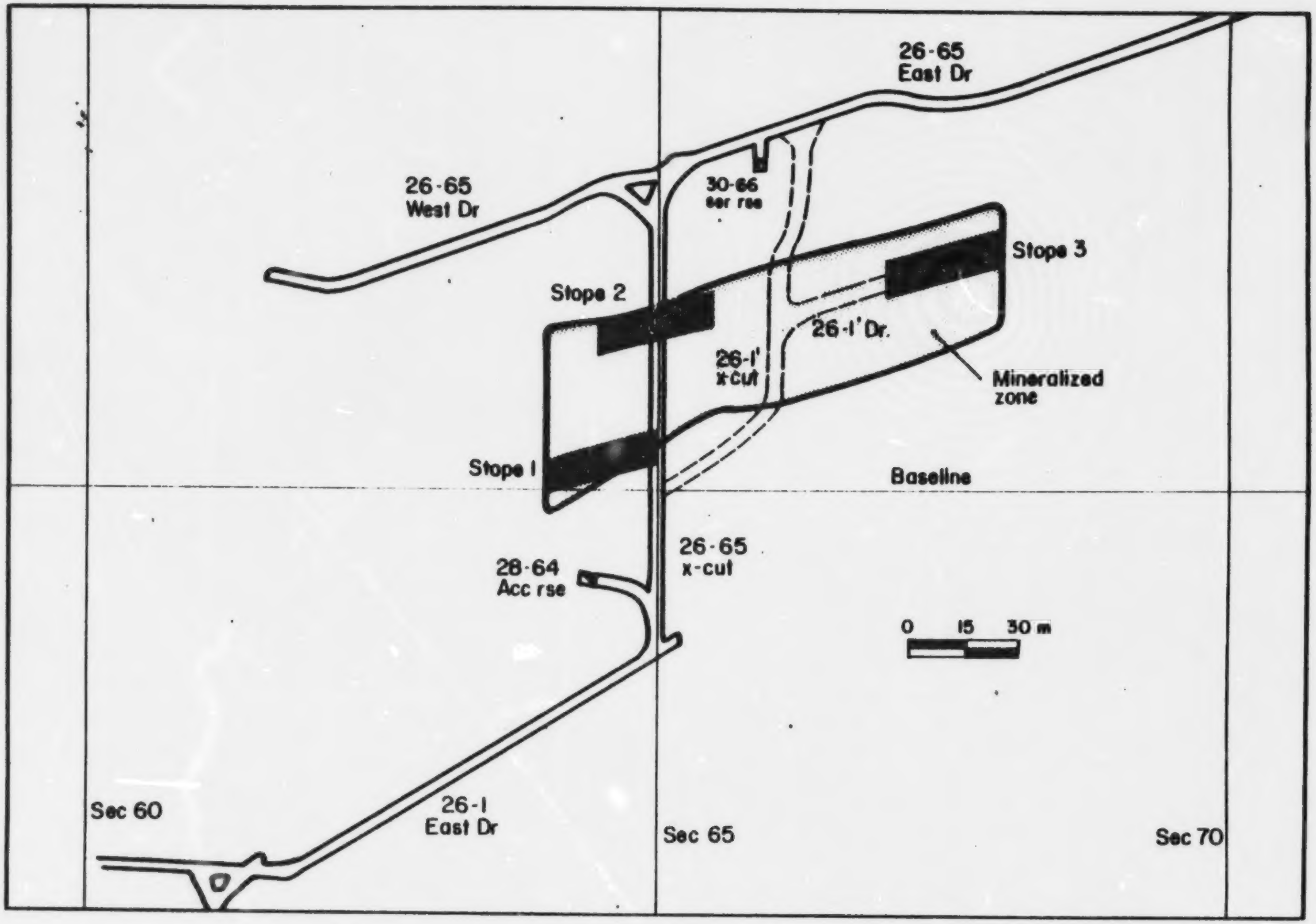

26-65
East Dr
26-65
West Dr
30-66
ser rse
Stope 3
Stope 2
26-1' Dr.
26-1'
x-cut
Mineralized
zone
Stope 1
Baseline
26-65
x-cut
28-64
Acc rse
0
15
30 m
Sec 60
26-1
East Dr
Sec 65
Sec 70

Figure 1 - Plan View of 2650 Level

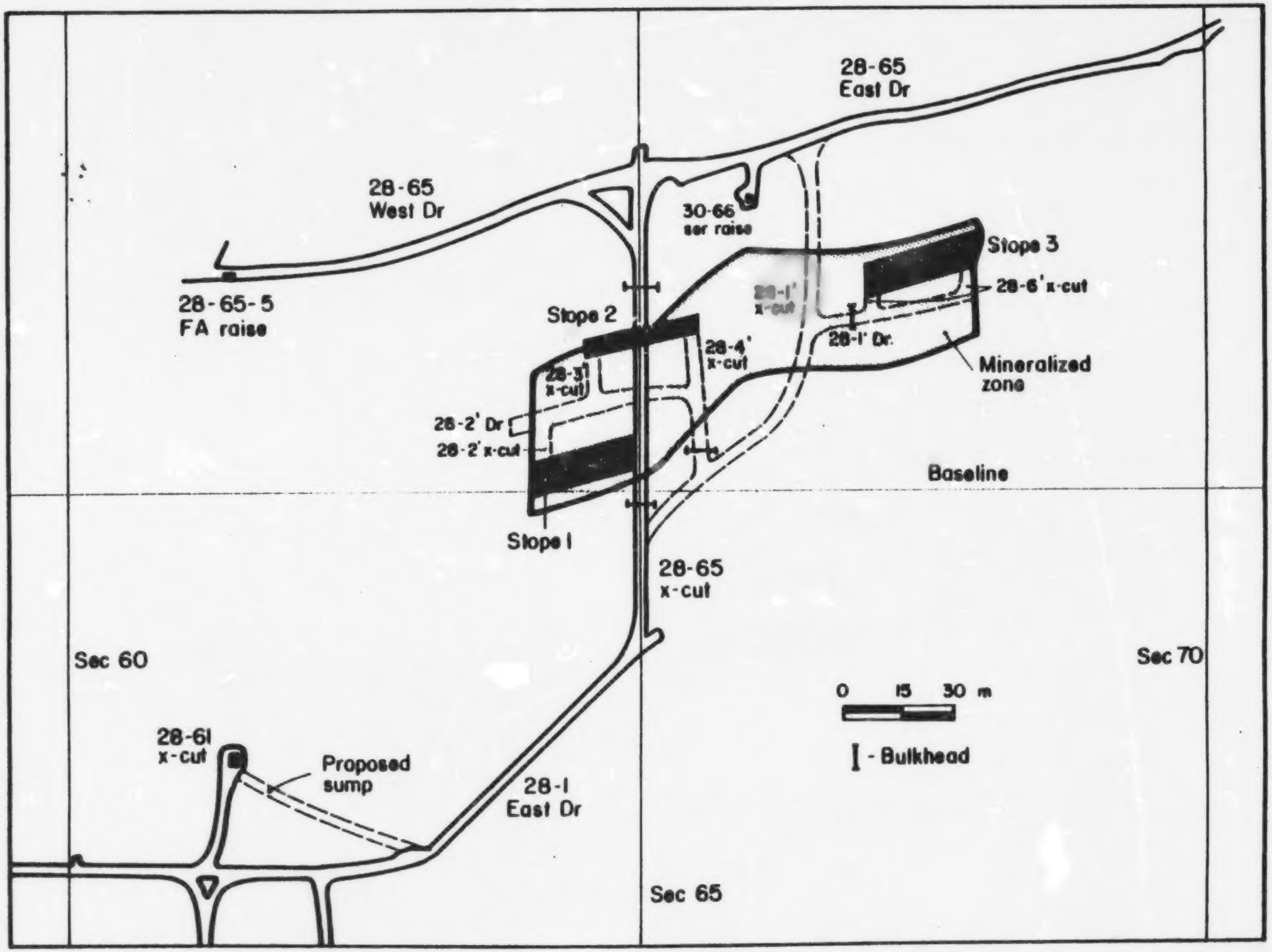

28-65
East Dr
28-65
West Dr
30-66
ser raise
Stope 3
28-6' x-cut
28-65-5
FA raise
Stope 2
28-1'
x-cut
28-4'
x-cut
28-1' Dr.
Mineralized
zone
28-3'
x-cut
28-2' Dr
28-2' x-cut
Baseline
Stope 1
28-65
x-cut
Sec 60
Sec 70
0
15
30 m
- Bulkhead
28-61
x-cut
Proposed
sump
28-1
East Dr
Sec 65

Figure 2 - Plan View of 2850 Level

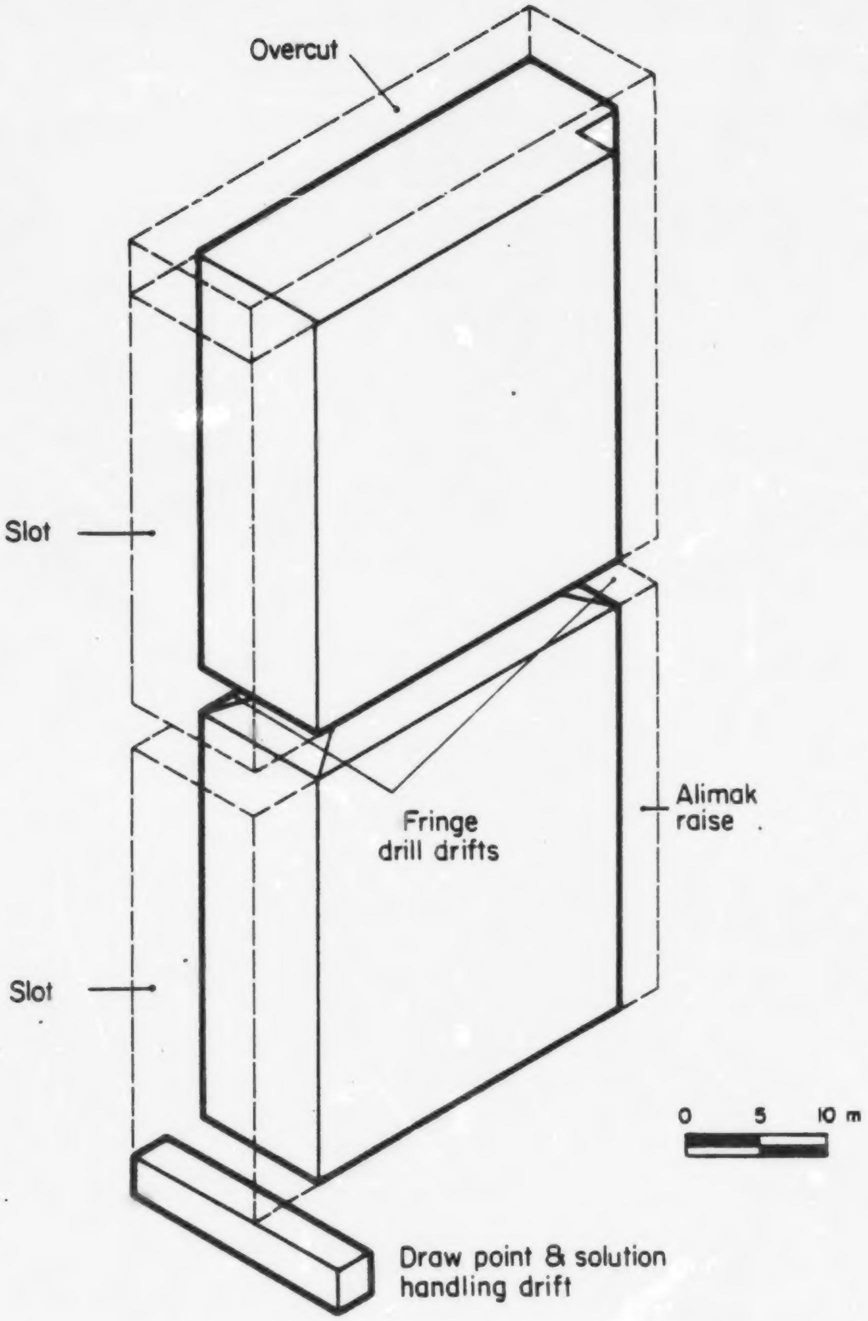

Overcut
Slot
Fringe drill drifts
Alimak raise
Slot
0 5 10 m
Draw point & solution handling drift

Figure 3 - Isometric View of Stopes 1 and 3

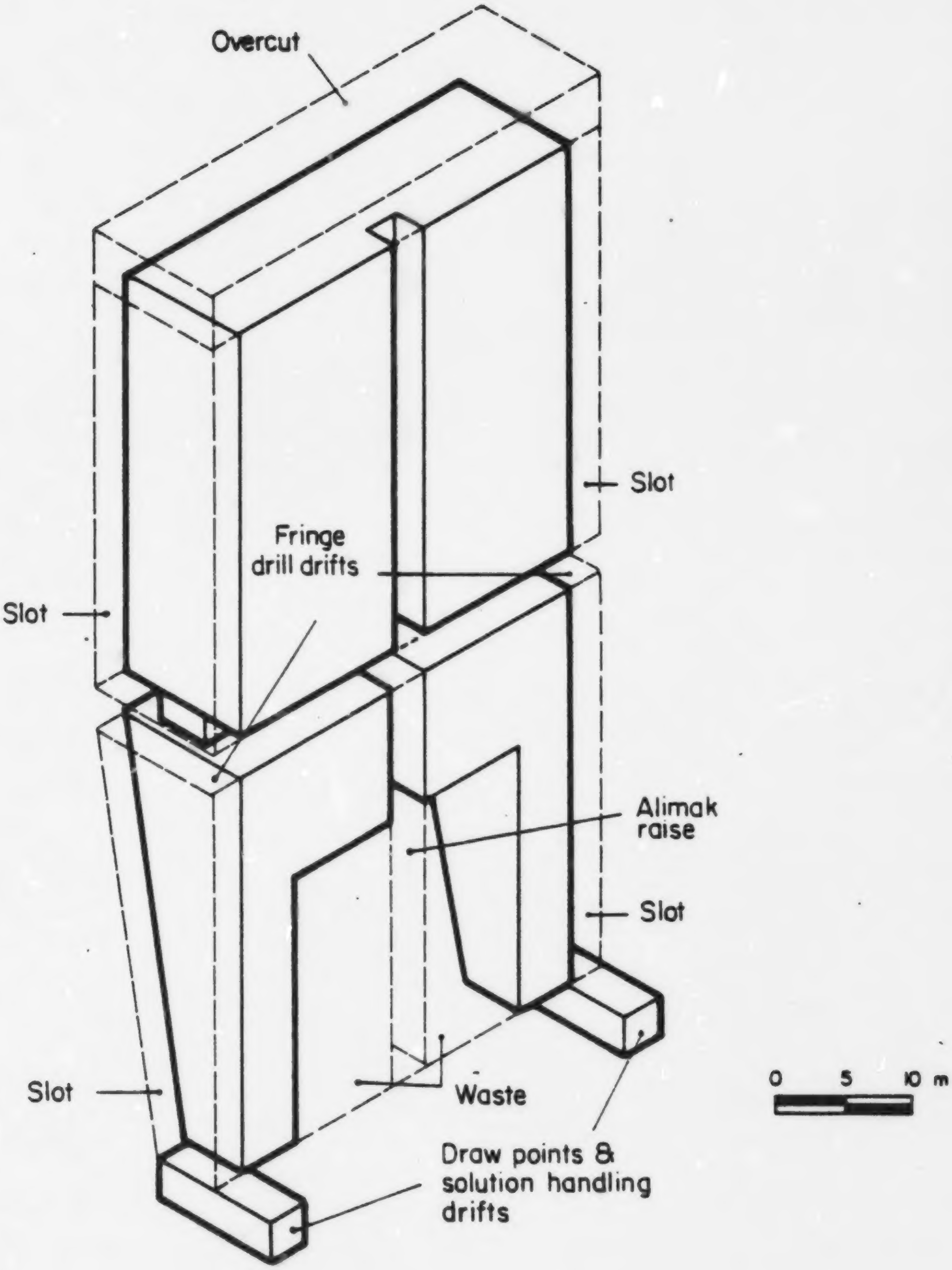

Overcut
Slot
Fringe drill drifts
Slot
Alimak raise
Slot
Slot
Waste
Draw points & solution handling drifts
0 5 10 m

Figure 4 - Isometric View of Stope 2

are required. This necessitates that the Alimak raise be driven in the middle of the stope from 28-65 crosscut.

All preparation included, about 25% of the stopes is removed to allow for swell. This is summarized in Table 2 and Table 3 and indicates the amount and grade of ore left for leaching. The ore mined during development will be processed and the resulting revenues will be credited to the development costs.

5.3.1 Overcuts

Overcuts spanning the entire width and length of the stopes are required for installation of the sprinkler system to be used for leaching. These are 3.7 m high and the back should be bolted at 2.7 m intervals alternating with 2 and 4 bolt patterns resulting in 6 rows of 2 and 5 rows of 4 bolts per stope.

5.3.2 Undercuts (Draw points)

Draw points measuring initially 2.7 m x 2.7 m are located on the western end of stopes 1 and 3 and on both ends of stope 2 for the full width of the stope. These will then be slashed to 4.6 m. Once the slot has been mucked out, the draw points and their access are to be used to handle the leaching solution.

5.3.3 Alimak Raises

Alimak raises are driven approximately 60 m from 2850 to 2650 level, to develop the 2750 sub-levels. The bottom part of the raise, from 2850 to 2750 will be used for mucking while the top part, from 2750 to 2650, will be used for access. These are located at the south eastern corners of stopes 1 and 3 and in the middle of the southern size of stope 2. 28-65 crosscut can be used to install the Alimak platforms and rails for stopes 1 and 2, whereas newly driven 28-6' crosscut is used in the case of stope 3.

TABLE 2

Summary of Stope Preparation

STOPE	ITEM	VOLUME (m3)	AMOUNT (tonnes)	GRADE (%)		TONNES OF COPPER	
				stope 1	stope 3	stope 1	stope 3
1 & 3	OVERCUT	1057.24	3020.68	0.90	1.00	27.19	30.21
	ALIMAK	414.80	1185.15	0.90	1.00	10.67	11.85
	SUBLEVEL						
	SOUTH	201.93	576.95	1.20	0.70	6.92	4.04
	NORTH	201.93	576.95	1.20	0.70	6.92	4.04
	EAST	46.66	133.30	1.40	1.10	1.87	1.47
	WEST	56.38	161.07	1.30	1.10	2.09	1.77
	DRAW PT.	113.02	322.92	0.40	1.00	1.29	3.23
	SLOT	2897.62	8278.92	1.10	1.00	91.07	82.79
		TOTAL:	14255.95			148.02	139.39
				AVERAGE GRADE:		1.04	0.98
2	OVERCUT	1057.24	3020.68		0.80		24.17
	ALIMAK	414.80	1185.15		0.40		4.74
	SUBLEVEL						
	SOUTH	201.93	576.95		1.00		5.77
	NORTH	201.93	576.95		0.70		4.04
	EAST	46.66	133.30		1.40		1.87
	WEST	26.24	74.98		1.00		0.75
	DRAW PT.	113.02	322.92		1.00		3.23
	SLOTS						
	EAST						
	<2750	579.76	1656.46		1.00		1.66
	TO 2850	423.84	1210.98		1.40		16.95
	WEST						
	<2750	579.76	1656.46		0.70		11.60
	TO 2850	423.84	1210.98		0.70		8.48
		TOTAL:	11625.82				83.24
				AVERAGE GRADE:			0.72

TABLE 3

Leachable Ore

STOPE	AMOUNT TO BE LEACHED (tonnes)	GRADE (%)	LEACH RECOVERY (%)	TOTAL COPPER (tonnes)
1	33993.29	0.80	55	149.57
2	26188.24	0.70	55	100.82
3	33993.29	0.90	55	168.27
TOTALS:	94174.83	0.80	n.a	418.66

TABLE 3: Leachable Ore

5.3.4 Fringe Drill Drifts and Crosscuts

Once access to the 2750 and 2650 sub-levels has been completed via Alimak raises, 2.7 m x 2.7 m drill drifts are driven around the perimeter of the stopes in the ore. The crosscut opening the slot area is slashed to 5.8 m for stopes 1 and 3.

5.3.5 Drilling and Blasting

Presently at Geco, a 3.4 m x 3.7 m blasting pattern of 114 mm diameter hole gives an average muck size of 400 mm. By reducing the burden to 1.5 m and the spacing to 3 m, the charge is increased by 2.8 times and therefore, should ensure an average muck size of less than 150 mm. The patterns required are shown in Figures 5 and 6. Here, no holes are within 1.2 m of the proposed stope wall. Because the overcut spans the entire stope, the top half of the stopes are to be benched, thereby reducing drill requirements and facilitating drill set-up. Table 4 summarizes the amount of drilling required.

Once drilled, the stopes will be blasted sequentially in the context of a single blast with ANFO in dry holes, and slurry in wet holes.

5.3.6 Slots

The slots are drilled with 114 mm diameter holes and will be sequentially blasted simultaneously with the slot drop raise. This will have to be done in two parts; between 2650 and 2750 levels, and between 2750 and 2850 levels with the latter being completed first. The slots are located as shown in Figures 3 and 4 and the amount to be removed is given in Table 2. A plan view of the slot drilling patterns is shown in Figure 7.

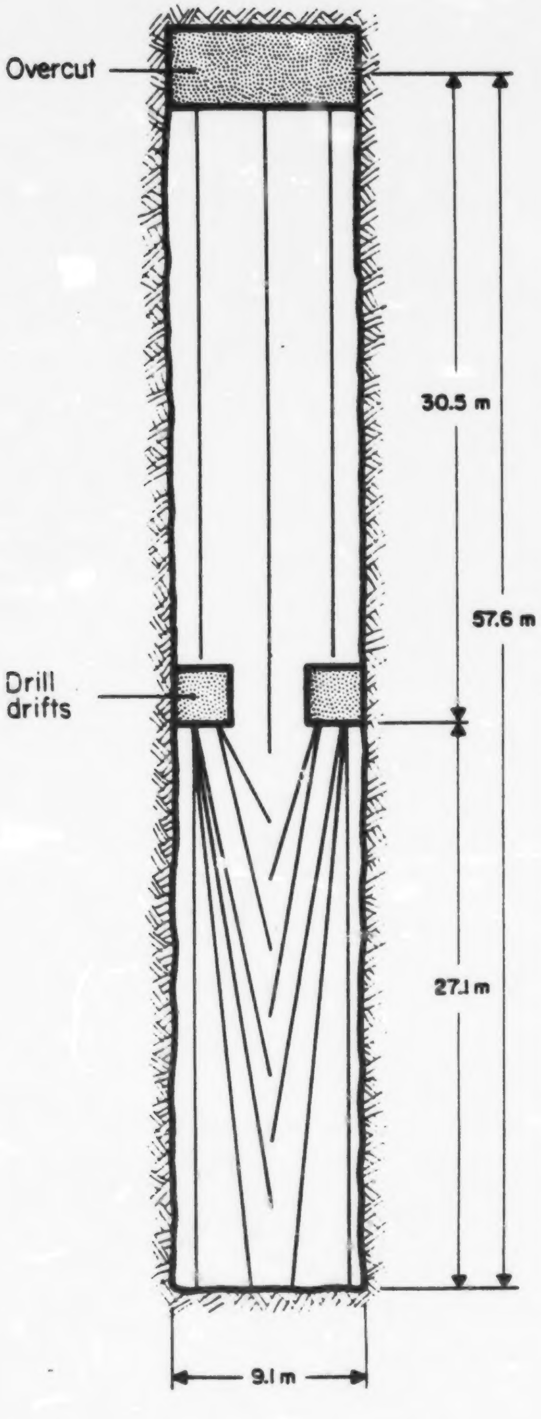
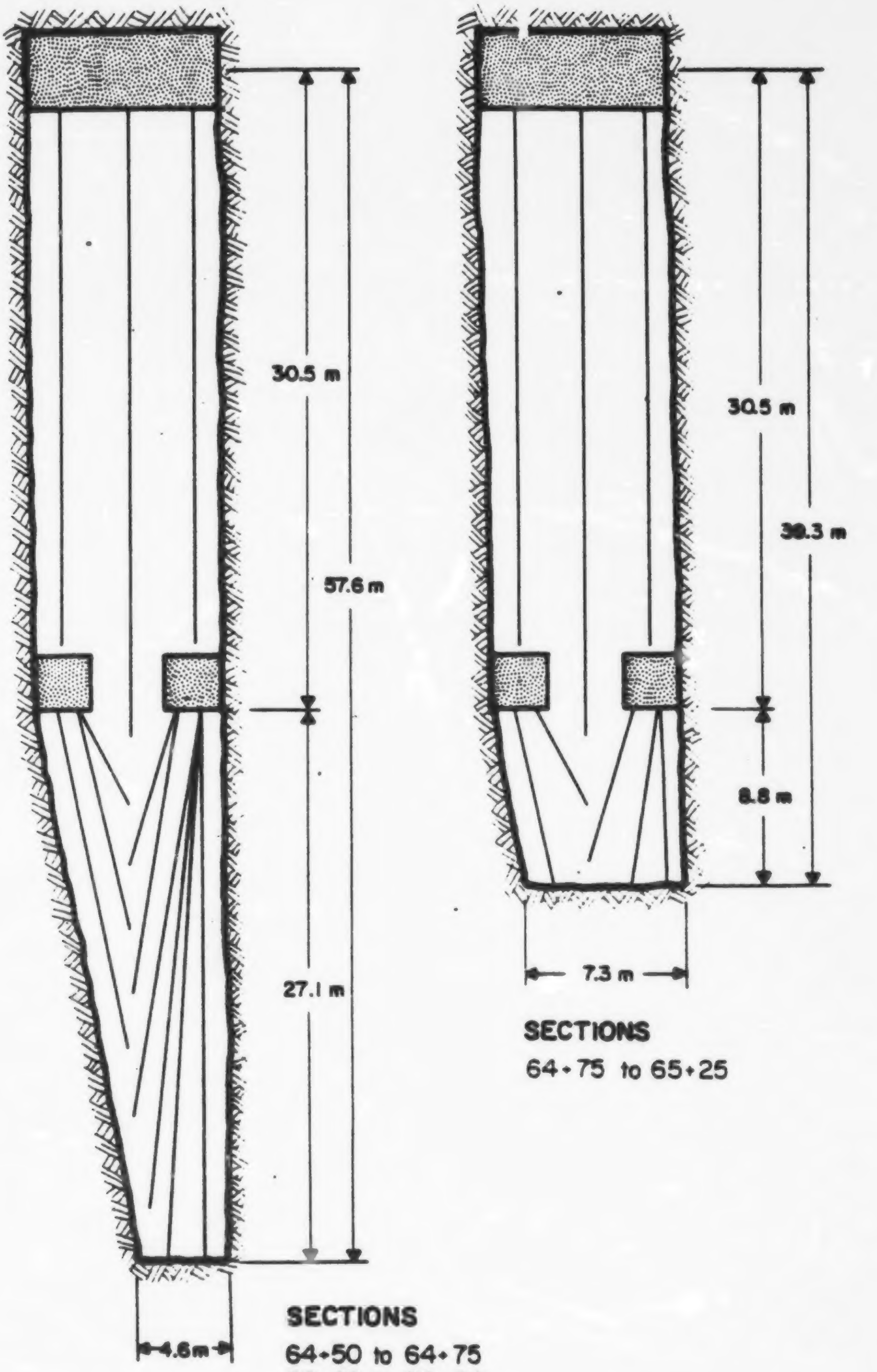

Overcut
Drill drifts
30.5 m
57.6 m
27.1 m
9.1 m

Figure 5 - Typical Drilling Pattern for Stopes 1 and 3


30.5 m
57.6 m
27.1 m
4.6m
30.5 m
39.3 m
8.8 m
7.3 m

SECTIONS
64+50 to 64+75
65+25 to 65+50

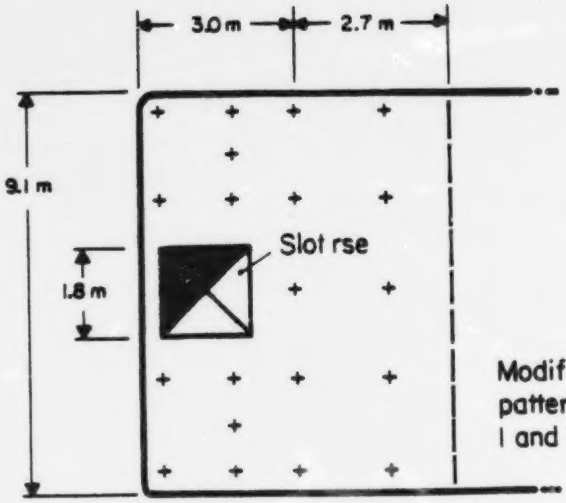
SECTIONS
64+75 to 65+25

Figure 6 - Typical Drilling Pattern for Stope 2

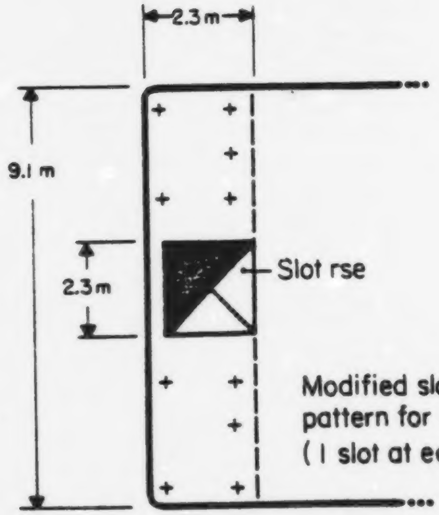
TABLE 4

Stope Drilling Required After Slot Has Been Drilled Off

STOPE	LOCATION	HOLES/ RING	# OF RINGS	TOTAL/ RING (m)	STOPE TOTAL (m)
1	<2750 BT. 64 + 00 AND 65 + 00	3	16	82	1312.00
	>2750 BT. 64 + 00 AND 65 + 00	11	16	199	3184.00
					SUB: 4496.00
2	<2750 BT. 64 + 50 AND 65 + 50	3	18	82	1476.00
	>2750 BT. 64 + 50 AND 64 + 75	9	2	154	308.00
	64 + 75 AND 65 + 25	5	10	40	400.00
	65 + 25 AND 65 + 50	9	2	154	308.00
					SUB: 2492.00
3	<2750 BT. 67 + 00 AND 68 + 00	3	16	82	1312.00
	>2750 BT. 67 + 00 AND 68 + 00	11	16	199	3184.00
					SUB: 4496.00
					TOTAL: 11484.00


3.0 m
2.7 m
9.1 m
1.8 m
Slot rse

Modified slot drilling pattern for stopes 1 and 3


2.3m
9.1 m
2.3m
Slot rse

Modified slot drilling pattern for stope 2 (1 slot at each end)

Figure 7 - Plan View of Slot Drilling Patterns

5.3.7 Bulkheads

Bulkheads are to be installed to contain the leachate in case of power failure or system breakdown. Four bulkheads are required on 2850 level (see Figure 2). In 28-65 crosscut, these will permanently seal the stoping zone on the north and south sides. Those in 28-4' crosscut and 28-1' east drift will be equipped with automatic valves which will automatically shut-off in case of pump stopage.

6. PERMEABILITY TESTING

An evaluation of rock mass permeability in the area of the Geco mine proposed in this investigation was carried out in existing boreholes by Geomines Ltd. during the month of February 1985. The study concluded that the permeability is very low (10^{-8} cm/sec) and therefore, that the loss of leachate would be insignificant. However, it is reported that the permeability could increase as a result of mining activity in the area. A copy of the report is included in Appendix A.

For the purpose of this study, it is assumed that 100% of the solution will be recovered.

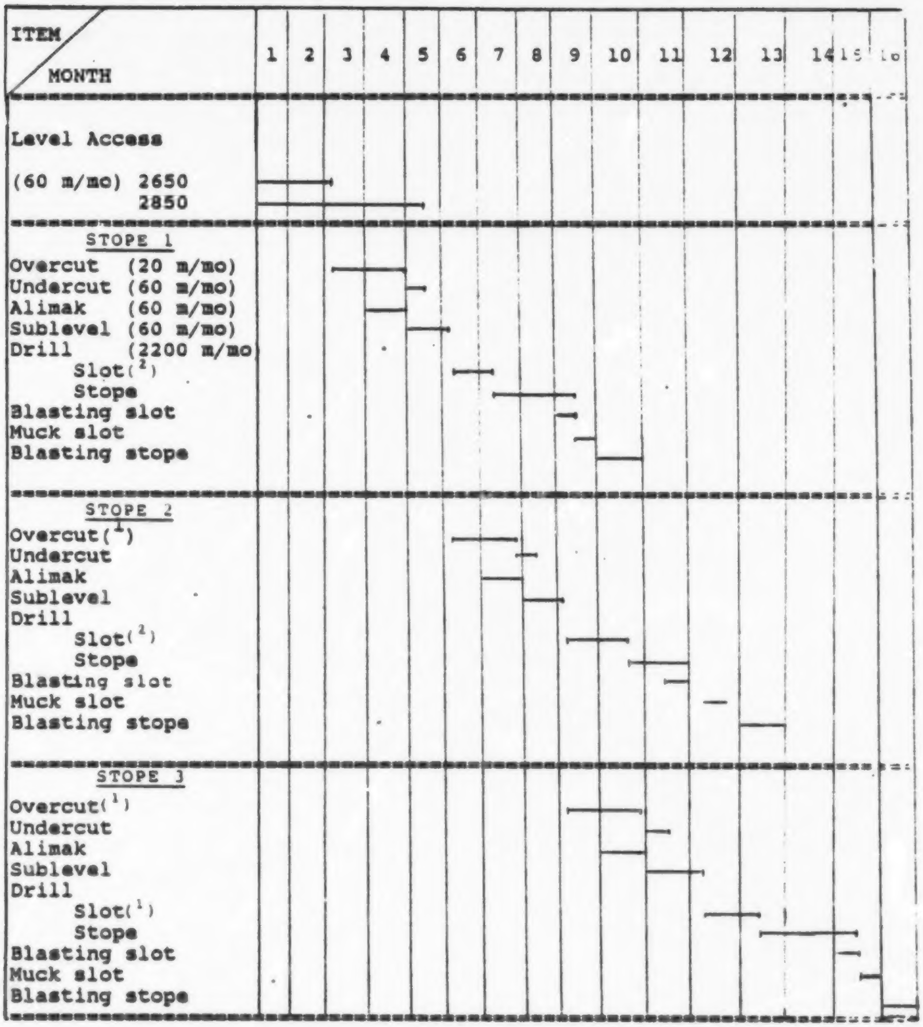
7. SCHEDULE

Table 5 outlines the schedule of this project based on the following assumptions:

- 2650 and 2850 level development can be carried out simultaneously;
- the undercut can be developed at the same time as the sub-level is for a given stope;
- slots can be blasted once drilling has progressed to the far side of the stope.

TABLE 5

Development and Blasting Schedule



(1) rates are the same as above.

(2) includes slot raise

The 2650 and 2850 stope access levels are initially developed to accommodate all three stopes, however, preparation of the first stope begins before this is completed. Once the overcut, undercut and sub-level development have been completed for a stope, this can immediately commence for the next stope. Hence, approximately a four month overlap between stopes exists. Therefore, it will take 16 months to get all three stopes ready for leaching.

8. COST SCHEDULE

Tables 6 through 9 outline the cost schedules for level access and the preparation of stopes 1, 2 and 3 respectively. The costs used for mining and processing were those given by Geco personnel and these are summarized in Table 10.

The total cost of developing the three stopes is about $1.39 x 10^6 representing $14.74/tonne of ore to be leached.

9. CONCLUSIONS

1. Mining costs incurred to prepare stopes for bacterially-assisted leaching are $14.74/tonne.

2. After stope preparation, an average of about 31,500 tonnes/stope at about 0.8% Cu remains for leaching.

3. Total revenues generated by processing development ore are almost $750,000 assuming a base case copper price for $2.20/kg.

4. The three stopes will be ready for leaching after 16 months.

5. Permeability testing showed that an insignificant loss of leachate can be expected.

TABLE 6

Cost Schedule for 2650 and 2850 Access

TASK / MONTH	1	2	3	4	5	6
METERAGE						
LEVEL ACCESS						
2650						
26-1'XCUT	61	33	18			
26-1' DR		28				
2850						
28-1'XCUT	61	44	19			
28-1' DR		17	26			
28-4'XCUT			16	22		
28-2' DR				39	10	
28-2'XCUT					8	
28-3'XCUT					10	
SUMP					33	28
MONTHLY TOTAL:	122	122	79	61	61	28
ORE PRODUCED (t):	516	1591	946	1311.5	473	0
COSTS (000'$)						
DRIFTING	80.85	80.85	52.36	40.43	40.43	18.56
MILLING	2.84	8.77	5.21	7.23	2.61	0.00
CONCENTRATE (t):	10.79	33.27	19.78	27.43	9.89	0.00
FREIGHT	0.42	1.28	0.76	1.06	0.38	0.00
SMELTING	1.55	4.77	2.84	3.93	1.42	0.00
COPPER (t):	2.67	8.25	4.90	6.80	2.45	0.00
REFINING	0.59	1.81	1.08	1.50	0.54	0.00
TOTAL:	86.25	97.49	62.25	54.14	45.37	18.56
GENERATED REVENUE:	5.89	18.17	10.81	14.98	5.40	0.00
NET COST (000'$):	80.35	79.31	51.44	39.16	39.97	18.56

TABLE 7

Cost Schedule for Stope Number 1

TASK / MONTH	1	2	3	4	5	6	7	8	9	10
STOPE PREPARATION										
METERAGE										
DRIFTING										
OVERCUT			14	17						
UNDERCUT				9						
SUBLEVEL					60	16				
SLASHING						9				
MONTHLY TOTAL (m):	0	0	14	26	60	25	0	0	0	0
COST (000'$):	0.00	0.00	31.28	43.95	39.78	16.58	0.00	0.00	0.00	0.00
ORE PRODUCED (t):	0.00	0.00	1363.60	1849.30	1290.00	537.50	0.00	0.00	0.00	0.00
ALIMAK RSE.				58						
COST (000'$):				65.08						
ORE PRODUCED (t):				978.00						
DRILLING										
SLOT RSE.						715				
SLOT						550	550			
STOPE							1100	2200	1196	
MONTHLY TOTAL (m):	0	0	0	0	0	1265	1650	2200	1196	0
COST (000'$):	0.00	0.00	0.00	0.00	0.00	18.51	24.14	32.19	17.50	0.00
BLASTING										
SLOT (t):									8279	
STOPE(t):										34558
COST (000'$):									20.20	84.32
MUCKING SLOT										
COST (000'$):									4.14	
ORE PRODUCED (t):									8279	
COSTS (000'$):										
TOTAL MINING COST:	0.00	0.00	31.28	109.03	39.78	35.08	24.14	32.19	41.84	84.32
HOISTING:	0.00	0.00	1.50	3.11	1.42	0.59	0.00	0.00	9.11	0.00
MILLING:	0.00	0.00	7.51	15.58	7.11	2.96	0.00	0.00	45.62	0.00
CONCENTRATE (t):	0.00	0.00	46.75	96.94	44.23	18.43	0.00	0.00	283.85	0.00
FREIGHT:	0.00	0.00	1.80	3.74	1.71	0.71	0.00	0.00	10.95	0.00
SMELTING:	0.00	0.00	6.70	13.89	6.34	2.64	0.00	0.00	40.68	0.00
COPPER (t):	0.00	0.00	11.59	24.02	10.96	4.57	0.00	0.00	70.34	0.00
REFINING:	0.00	0.00	2.55	5.28	2.41	1.00	0.00	0.00	15.47	0.00
TOTAL:	0.00	0.00	51.35	150.63	58.76	42.99	24.14	32.19	163.66	84.32
GENERATED REVENUE:	0.00	0.00	25.54	52.96	24.16	10.07	0.00	0.00	155.07	0.00
NET COST (000'$):	0.00	0.00	25.81	97.68	34.60	32.92	24.14	32.19	8.60	84.32

TABLE 3

Cost Schedule for Stope Number 2

TASK / MONTH	4	5	6	7	8	9	10	11	12	13
STOPE PREPARATION										
METERAGE										
DRIFTING										
OVERCUT			12	19						
UNDERCUT				3	6					
SUBLEVEL					54	22				
SLASHING										
MONTHLY TOTAL (m):	0	0	12	22	60	22	0	0	0	0
COST (000'$):	0.00	0.00	26.81	44.44	39.78	14.59	0.00	0.00	0.00	0.00
ORE PRODUCED (t):	0.00	0.00	1168.80	1915.10	1290.00	473.00	0.00	0.00	0.00	0.00
ALIMAK RSE.				58						
COST (000'$):				65.08						
ORE PRODUCED (t):				978.00						
DRILLING										
SLOT RSE.						1430				
SLOT							1100			
STOPE							1100	1392		
MONTHLY TOTAL (m):	0	0	0	0	0	1430	2200	1392	0	0
COST (000'$):	0.00	0.00	0.00	0.00	0.00	20.92	32.19	20.36	0.00	0.00
BLASTING										
SLOT								5732		
STOPE					26753					26753
COST (000'$):					65.28			13.99		65.28
MUCKING SLOT										
COST (000'$):									2.87	
ORE PRODUCED (t):									5732	
COSTS (000'$):										
TOTAL MINING COST:	0.00	0.00	26.81	109.52	105.06	35.51	32.19	34.35	2.87	65.28
HOISTING:	0.00	0.00	1.29	3.18	1.42	0.52	0.00	0.00	6.31	0.00
MILLING:	0.00	0.00	6.44	15.94	7.11	2.61	0.00	0.00	31.58	0.00
CONCENTRATE (t):	0.00	0.00	28.85	71.42	31.94	11.68	0.00	0.00	141.50	0.00
FREIGHT:	0.00	0.00	1.11	2.76	1.23	0.45	0.00	0.00	5.46	0.00
SMELTING:	0.00	0.00	4.13	10.23	4.56	1.67	0.00	0.00	20.28	0.00
COPPER (t):	0.00	0.00	7.15	17.70	7.89	2.89	0.00	0.00	35.06	0.00
REFINING:	0.00	0.00	1.57	3.89	1.74	0.64	0.00	0.00	7.71	0.00
TOTAL:	0.00	0.00	41.36	145.53	121.11	41.39	32.19	34.35	74.20	65.28
GENERATED REVENUE:	0.00	0.00	15.76	39.02	17.40	6.38	0.00	0.00	77.30	0.00
NET COST (000'$):	0.00	0.00	25.60	106.51	103.72	35.01	32.19	34.35	-3.10	65.28

TABLE 9

Cost Schedule for Stope Number 3

TASK / MONTH	7	8	9	10	11	12	13	14	15	16
STOPE PREPARATION										
METERAGE										
DRIFTING										
OVERCUT			13	18						
UNDERCUT				6	3					
SUBLEVEL					57	19				
SLASHING						9				
MONTHLY TOTAL (m):	0	0	13	24	60	28	0	0	0	0
COST (000'$):	0.00	0.00	29.05	44.20	39.78	18.56	0.00	0.00	0.00	0.00
ORE PRODUCED (t):	0.00	0.00	1266.20	1882.20	1290.00	602.00	0.00	0.00	0.00	0.00
ALIMAK RSE.				58						
COST (000'$):				65.08						
ORE PRODUCED (t):				978.00						
DRILLING										
SLOT RSE.						715				
SLOT						473	627			
STOPE							1562	2200	734	
MONTHLY TOTAL (m):	0	0	0	0	0	1188	2189	2200	734	0
COST (000'$):	0.00	0.00	0.00	0.00	0.00	17.38	32.03	32.19	10.74	0.00
BLASTING										
SLOT (t):								8279		
STOPE(t):										34558
COST (000'$):								20.20		84.32
MUCKING SLOT										
COST (000'$):									4.14	
ORE PRODUCED (t):									8279	
COSTS (000'$):										
TOTAL MINING COST:	0.00	0.00	29.05	109.27	39.78	35.94	32.03	52.39	14.88	84.32
HOISTING:	0.00	0.00	1.39	3.15	1.42	0.66	0.00	0.00	9.11	0.00
MILLING:	0.00	0.00	6.98	15.76	7.11	3.32	0.00	0.00	45.62	0.00
CONCENTRATE (t):	0.00	0.00	43.41	98.06	44.23	20.64	0.00	0.00	283.85	0.00
FREIGHT:	0.00	0.00	1.67	3.78	1.71	0.80	0.00	0.00	10.95	0.00
SMELTING:	0.00	0.00	6.22	14.05	6.34	2.96	0.00	0.00	40.68	0.00
COPPER (t):	0.00	0.00	10.76	24.30	10.96	5.11	0.00	0.00	70.34	0.00
REFINING:	0.00	0.00	2.37	5.35	2.41	1.13	0.00	0.00	15.47	0.00
TOTAL:	0.00	0.00	47.68	151.36	58.76	44.80	32.03	52.39	136.70	84.32
GENERATED REVENUE:	0.00	0.00	23.72	53.57	24.16	11.28	0.00	0.00	155.07	0.00
NET COST (000'$):	0.00	0.00	23.96	97.79	34.60	33.53	32.03	52.39	-18.36	84.32

TABLE 10

Unit Costs Used in the Analysis

UNIT COSTS ($)								
DRIFTING (/m)	ALIMAK (/m)	DRILLING (/m)	BLASTING (/t)	MUCKING (/t)	HOISTING (/t)	MILLING (/t)	FREIGHT (/t)	SMELTING/REFINING (/kg)
663.00	1122.00	14.63	2.44	0.50	1.10	5.51	38.58	.83

APPENDIX A

EVALUATION OF ROCK MASS
PERMEABILITY FOR A PROPOSED
IN SITU LEACHING TEST

GEOMINES LTEE LTD.

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NORANDA RESEARCH CENTER

Quebec

EVALUATION OF ROCK MASS PERMEABILITY FOR A PROPOSED IN SITU LEACHING TEST

February 20, 1985

REF. GEOMINES 85192

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RÉSUMÉ

Le Centre de Recherche Noranda a retenu les services de Géomines Ltée pour évaluer la perméabilité du massif rocheux dans le cadre d'un projet de lixiviation en place à la Mine Geco à Manitouwadge, Ontario.

Cette évaluation a été faite à l'aide d'essais d'injection dans des forages d'exploration et par l'observation des conditions hydrogéologiques dans la mine.

Les conclusions de l'étude sont que la perméabilité du massif rocheux est très faible (10^{-8} cm/sec) dans la zone d'essai proposée et que les pentes de lixiviats au travers des parois ne seront pas importantes; des pertes pourraient cependant se produire dû aux activités de minage (forages, sautage et fracturation du massif).

ABSTRACT

Geomines Ltd. was retained by Noranda Research Center to evaluate the rock mass permeability for a proposed in situ leaching test at the Geco Mine in Manitouwadge, Ontario.

This evaluation was performed through injection testing of exploration boreholes and observation of hydrogeological conditions in the mine. The conclusion of this study is that the rock mass permeability is very low (10^{-8} cm/sec) in the proposed test zone and that loss of leachate through natural permeability would be insignificant; the main problems that could occur are related to mine induced permeability (drilling, fractures).

1. INTRODUCTION

1.1 Mandate

On January 3rd, 1985, following a proposal submitted by telex on December the 21st, 1984, GEOMINES LTD. was retained by the MINING TECHNOLOGY DIVISION of the NORANDA RESEARCH CENTER to perform a hydrogeological evaluation at the Geco Division of Noranda Mines. The purpose of this study was to determine the rock mass permeability in a area of the mine where a leaching experiment is planned in a stope. This study was to be completed before the 22nd of February, 1985.

1.2 Background information

The zone to be investigated at Geco contains fairly large volumes of low grade ore which cannot be mined economically at present. The purpose of the test is to break up this ore inside a stope and to proceed with leaching so as to evaluate the economic and technical feasibility of this type of in situ leaching. This experiment is to be undertaken by the Geco Division with the joint cooperation of Noranda Research and CANMET. The purpose of the hydrogeologic evaluation is to determine the amount of leachate which could be lost from the stope in the course of this experiment.

1.3 Work performed

The hydrogeologic evaluation was planned to include straddle-packer injection tests in two boreholes in addition to geological and hydrogeological surveys in the zone of the tests. Because of time constraints, the planned test holes (BQ size) could not be drilled and it was therefore decided to test existing EX size exploration holes with a single packer at the collar. This testing and the complementary underground surveys were performed in the week of the 11th to the 14th of February, 1985 by Mr. A. Liard, senior geologist at Geomines Ltd., who also prepared this report.

The performance of this study was greatly helped by the active collaboration of the Geco Division Geology department in the persons of Mr. R. B. Friesen, chief geologist, and Mr. F. Bakker, geologist, who provided background information and technical assistance, and Mr. G. Multamaki, engineer, who provided valuable assistance and participated in all underground work.

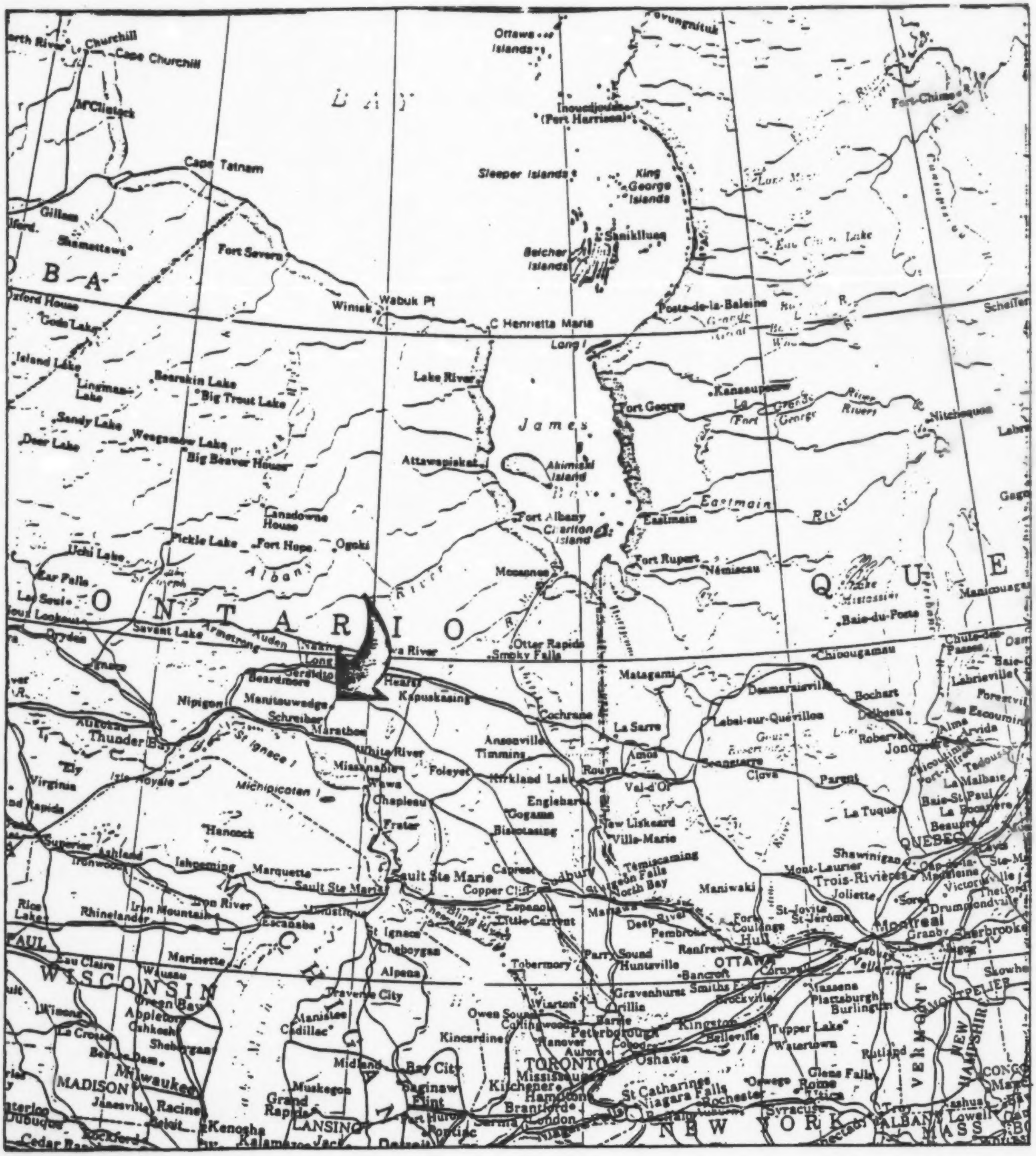
2. SITE DESCRIPTION

2.1 Location and access

The Geco Mine is located near the town of Manitouwadge in Northwestern Ontario and has been in continuous production since 1957 (see location map). Access to Manitouwadge by road is possible by travelling on the TransCanada Highway on the North shore of Lake Superior with a turnoff into highway 614 North, approximately 60 km East of Marathon. Alternatively, daily passenger plane service is now available from Toronto via Austin Airways.

2.2 Geologic setting

The main Geco orebody is a massive sulphide deposit which is interpreted as being of volcanogenic origin. High grade regional metamorphism and tectonic deformation have erased primary textural or structural features. The orebody and host rocks have been metamorphosed to the amphibolite facies. In a simplistic fashion, three main rock units are encountered: a grey gneiss in the structural foot wall; a sericite schist which contains massive and disseminated sulphides; a granitic gneiss in the structural hanging wall. These conformable rock units are steeply dipping and are crosscut by pegmatitic and diabase dykes of different ages. The sericite schist unit has textures which range from a mica schist to a sillimanite gneiss and contains numerous conformable quartz diorite dykes in addition to the massive sulphides. Summary observation indicates that gneissic textures predominate in the schist unit in the area of study (East of main ore zone).



Scale: 1:10,000,000

Location map

Faulting normal to the orebody is documented by mineral exploration and jointing of similar orientation can be observed in drifts. Also abundant jointing and minor shear zones parallel to foliation are noted. The rock types described and observed have intrinsically low permeabilities and porosities so that any permeability of the rock mass will be conferred by fractures.

2.3 Underground mine layout

The steep plunge of the orebody has allowed mining by the bulk blasthole method for most of the orebody. Access to the mine workings is by two surface shafts which go down to depths of 760 m (2500 feet) (no. 1 shaft) and 1235 m (4050 feet) (no 4 shaft) and which are connected to horizontal drifts at 60 m (200 feet) intervals. Ground control is assured in drifts by rock bolting and in ore stopes by filling with waste rock and cemented mill tailings. Most of the mining has taken place in the main orebody, but two parallel orebodies (the 4/2 copper zone and the 8/2 zinc zone) have seen some mining activity. In the vicinity of the proposed leaching test, the main orebody is not being mined, but development work and some mining have been done in the 4/2 copper zone.

2.4 Geometry of leaching test

The proposed leaching test will be performed in blasthole stopes which will be excavated in the main ore zone. These stopes are to be located between sections 65+00 and 69+00 (approximately 100 m East at no. 4 shaft) and between levels 2650 and 2850. Access to the test zone is by 2650 East and 2850 East drifts.

No mining has been undertaken in the main ore zone within 100 m of the test zone and the nearest mining has been in the 4/2 orebody below the 3050 level, i.e. approximately 60 m downdip and 30 m offset from the test zone. Some problems with ground control were experienced in the upper part of these last stopes.

The leaching test stopes are planned to be approximately 60 m high and 20 m deep and to be spread over 100 m along strike.

Leaching is to be done by trikling acidic water through the broken ore so that no hydraulic head will exist at the bottom of the stope.

3. EVALUATION OF ROCK MASS PERMEABILITY

3.1 Description of test boreholes

A total of five boreholes were used to test the permeability of the rock mass. These boreholes were drilled in EX size (3.77 cm diameter) from the 26-65 East drift for mineral exploration purposes. They are located from section 65+50 to 68+50 and dip downward at angles of 15 to 35°. The length of the boreholes varies from 73 to 133 m so that they crosscut the whole sericite schist unit and sometimes penetrate a significant amount into the grey gneiss unit. The drawings entitled «Geological section of tested zone» and «Plan view of test zone» annexed with this report show the location of these holes.

3.2 Testing method

The test method used was the so-called Lugeon test or water injection test. In this case, a mechanically tightened rubber sleeve packer was installed at the collar of the hole and water was injected into the borehole at constant pressure using an air powered drilling pump. Pressures were monitored with a manometer placed at the hole collar and flow was measured through changes in level in the water reservoir. The injection tests were performed at pressures of 500 KPa during thirty minutes.

The results of these tests can be interpreted as to rock mass permeability by dividing flow rate by the pressure head and a cavity constant which is dependent upon hole length and diameter.

Certain conditions must be assumed to exist before the test results can be used: the interpretation assumes that no single fracture causes the flow from the borehole but that multiple fractures do so; the borehole walls and intersected fractures are clean and free of drilling mud; the flow into the fractures is laminar near the borehole wall. One can presume that the first assumption is valid but it is doubtful that the other two assumptions are completely valid with the result that calculated permeabilities will tend to be lower than the real values.

3.3 Test results

The five injection tests performed give very similar results with very low flows (200-300 cm³ over 30 minutes) at relatively high pressures (500 KPa). The data from each test can be found annexed to this report while the results are summarized in table 1.

Table 1: Results of Injection Tests

Test Hole	Location Section #	Length (m)	Dip	Apparent * Permeability (cm/sec)
26-228	65 + 50	76	-26°	6×10^{-5}
26-248	65 + 50	79	-15°	4×10^{-9}
26-181	67 + 00	133	-25°	4×10^{-9}
26-184	68 + 00	99	-25°	5×10^{-9}
26-256	68 + 50	73	-35°	7×10^{-9}

*Remark: Water losses at the pump or in hose connections would render these values pessimistic.

These permeabilities are all of the order of 10^{-9} to 10^{-8} cm/sec which is extremely low for the equipment used. Water losses during testing and the limits of the method could easily cause the true permeabilities to be an order of magnitude more or less than the computed test results.

3.4 Complementary information

In view of the fact that limited testing could be performed in the time available, additionnal information was obtained on the ground water regime of the mine. This information consisted of mine pumping history, observation of underground structural features and observation of untested holes.

According to the Geco Geology department, no significant inflows were recorded when the 26-65 East drift was driven. This drift extends 600 m (2000 feet) to the 85+00 section. In the vicinity of the study zone, two faults and a diabase cross the drift at sections 66+30, 69+10 and 70+05, no evidence of flow along these structures and other parallel joints can be observed along the drift or in the 28-65 East drift at a lower level.

Along the 26-65 cross drift which traverses the sericite schist unit, numerous joints or minor shear zones parallel to foliation can be observed which exhibit no apparent water seepage.

Some of the untested boreholes in drifts 26-65 East (sections 68+00 and 69+00) and 28-65 East (holes 28-255 and 28-256) exhibit minor seepage into the drift (traces of limonite and other oxydes on the wall with minor dripping). An evaluation of the flow rates and hydraulic conditions at these holes gives permeabilities of the order of 10^{-7}cm/sec. Other holes where no flow is evident would give lower values of permeability still.

Observation of stope development at the 30-50 level shows no evidence of ground water inflow even in regions where mining activity has deteriorated ground conditions and requires heavy bolting.

4. CONCLUSION AND RECOMMENDATIONS

4.1 Permeability of rock mass

The results of the injection tests combined with observation data indicate that the sericite schist unit has very low permeability in the area of study. The geological nature of this rock unit and the depth of the test zone below surface support the test data in that this rock has very low intrinsic permeability and that the fractures which would allow water movement are closed by the ground pressure. Since no open fractures can be observed in the vicinity of the test zone, the permeability (hydraulic conductivity) of the rock mass can be stated to be of the order of 10^{-8} cm/sec. These values are representative of an unfractured igneous metamorphic rock and are at the limit of the testing methods used.

4.2 Evaluation of leakage from leaching stope

The volume of leachate which could seep into the rock from the test stopes can be evaluated from different assumptions on hydraulic gradients, infiltration area and rock mass permeability.

For infiltration to occur into the rock mass, water must be flowing away from the stope; since this stope is deep underground, it could easily act as a drain to the rock mass rather than drain into the rock. For flow into the rock to occur, there must be other mining developments at a lower elevation which drain the rock mass. Since some stopes do exist below the test zone, we will assume that a vertical hydraulic gradient (i) of unit value (1) exists below the test zone. The infiltration area (A) will be assumed to equal the area of the floor of the planned test stope, 20 m x 100 m, The permeability (K) of the rock mass will be as described above, i.e. 1×10^{-8} cm/sec.

From the above assumptions the flow (Q) can be computed (Darcy's law) as $Q = KiA = 2\ cm^3/sec$ or 17 l/day. The assumptions regarding the hydraulic gradient can be described as pessimistic with the result that the calculated flow rate can be regarded as conservative.

4.3 Possible difficulties

The preceding hydrogeological evaluation is concerned with an intact rock mass which may not be the case. In fact, exploration activity has produced numerous boreholes which cross the test zone. In addition, mining activity in the 4/2 ore zone, if continued to the same level as the test zone, could cause fractures to open in the rock mass. These two factors (open boreholes and/or fractures) could cause large changes in the rock mass permeability and could possibly cause significant loss of leachate. It is therefore advisable that steps be taken to ensure that all boreholes are properly sealed and that mining activity has minimal effect on the rock mass.

APPENDIX

- TEST DATA
- DRAWINGS

ESSAIS DE PERMEABILITE DANS LE ROCHER

orage: 26-184 Cote: ______ Hauteur d'appareil 0 Date 13/02/1985

rofondeur N.P. 0 A ______ hres Diamètre 3.77 cm Longueur d'essai 100 m

rofondeur bturateur upérieur	Temps		Pression manomètre	Lecture débit		Q/sec	Pression effective	K	Remarques
	Début	Fin		Début	Fin				
	0	30	500 KPA	0	± 330 cm^3	1.9x10^{-4}	500 KPA	5x10-9	

ESSAIS DE PERMEABILITE DANS LE ROCHER

orage: 26-181 Cote: ______ Hauteur d'appareil 0 Date 12/02/1985

rofondeur N.P. 0 A ______ hres Diamètre 3.77 cm Longueur d'essai 132 m

rofondeur oturateur upérieur	Temps		Pression manomètre	Lecture débit		Q/sec	Pression effective	K	Remarques
	Début	Fin		Début	Fin				
	0	30 min	~500 KPA	0	330 cm³	$1.9x10^{-4}$	500 KPA	$3.9x10^{-9}$	

GEOMINES LTEE

ESSAIS DE PERMEABILITE DANS LE ROCHER

rage: 26-256 Cote: ______ Hauteur d'appareil 0 Date 13/02/1985

rofondeur N.P. 0 A ______ hres Diamètre 3.77 cm Longueur d'essai 73 m

rofondeur oturateur ipérieur	Temps		Pression manomètre	Lecture débit		Q/sec	Pression effective	K	Remarques
	Début	Fin		Début	Fin				
	0	30	500 KPA	0	~330	$1.9x10^{-4}$	500 KPA	$7x10^{-9}$	

GEOMINES LTEE

ESSAIS DE PERMEABILITE DANS LE ROCHER

orage: 26-228 Cote: ______ Hauteur d'appareil 0 Date 13/02/1985

rofondeur N.P. 0 A ______ hres Diamètre 3.77 cm Longueur d'essai 76 m

rofondeur bturateur upérieur	Temps		Pression manomètre	Lecture débit		Q/sec	Pression effective	K	Remarques
	Début	Fin		Début	Fin				
	0	30	500 KPA	0	333 cm^3	1.9x10^{-4}	500 KPA	6x10^{-9}	

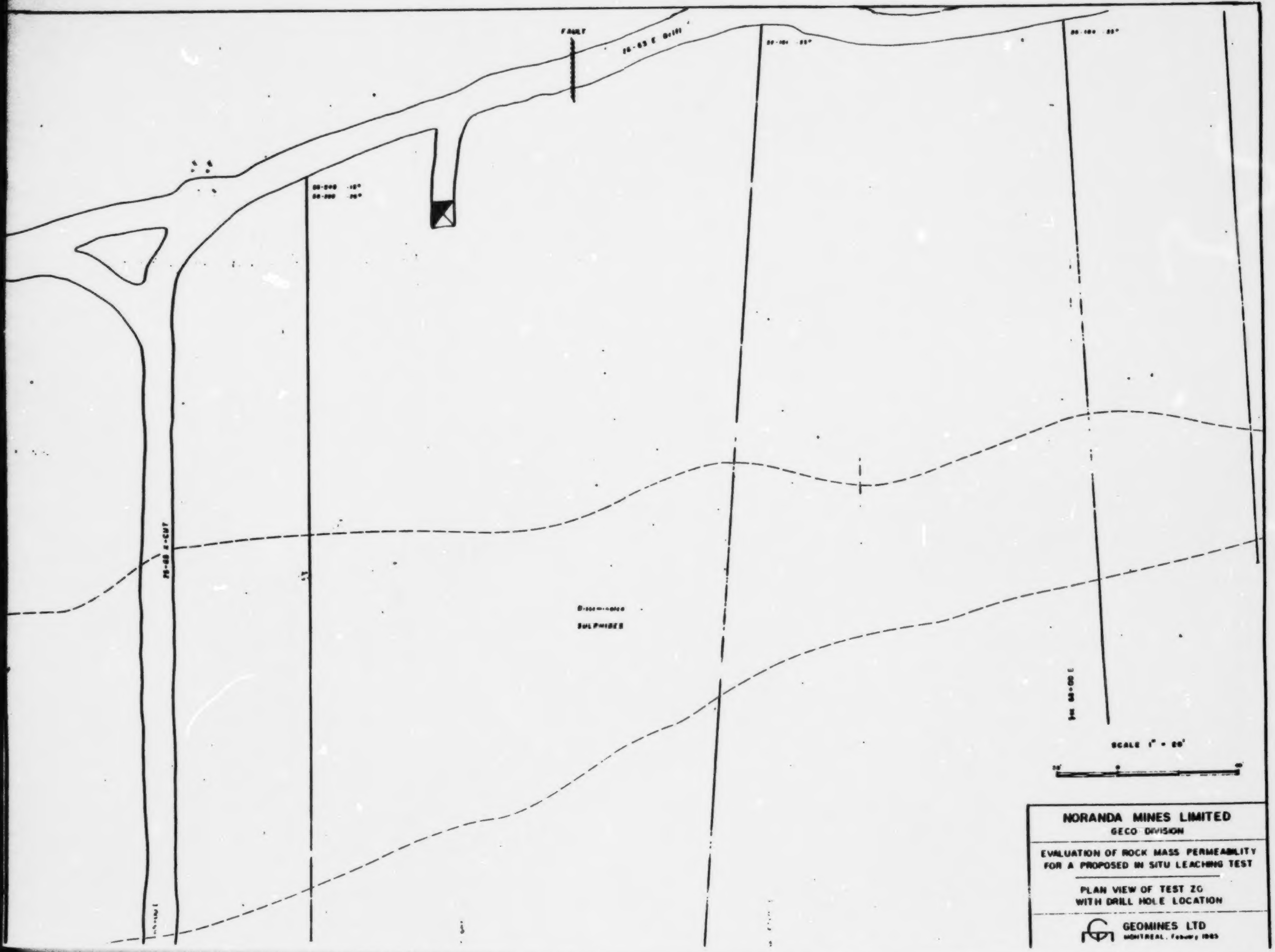
GEOMINES LTEE

ESSAIS DE PERMEABILITE DANS LE ROCHER

rage: 26-248 Cote: ______ Hauteur d'appareil 0 Date 13/02/1985

ofondeur N.P: 0 A ______ hres Diamètre 3.77 cm Longueur d'essai 79.8 m

ofondeur turateur périeur	Temps		Pression manomètre	Lecture débit		Q/sec	Pression effective	K	Remarques
	Début	Fin		Début	Fin				
	0	30	500 KPA	0	220	$1.2x10^{-4}$	500 KPA	$4x10^{-9}$	



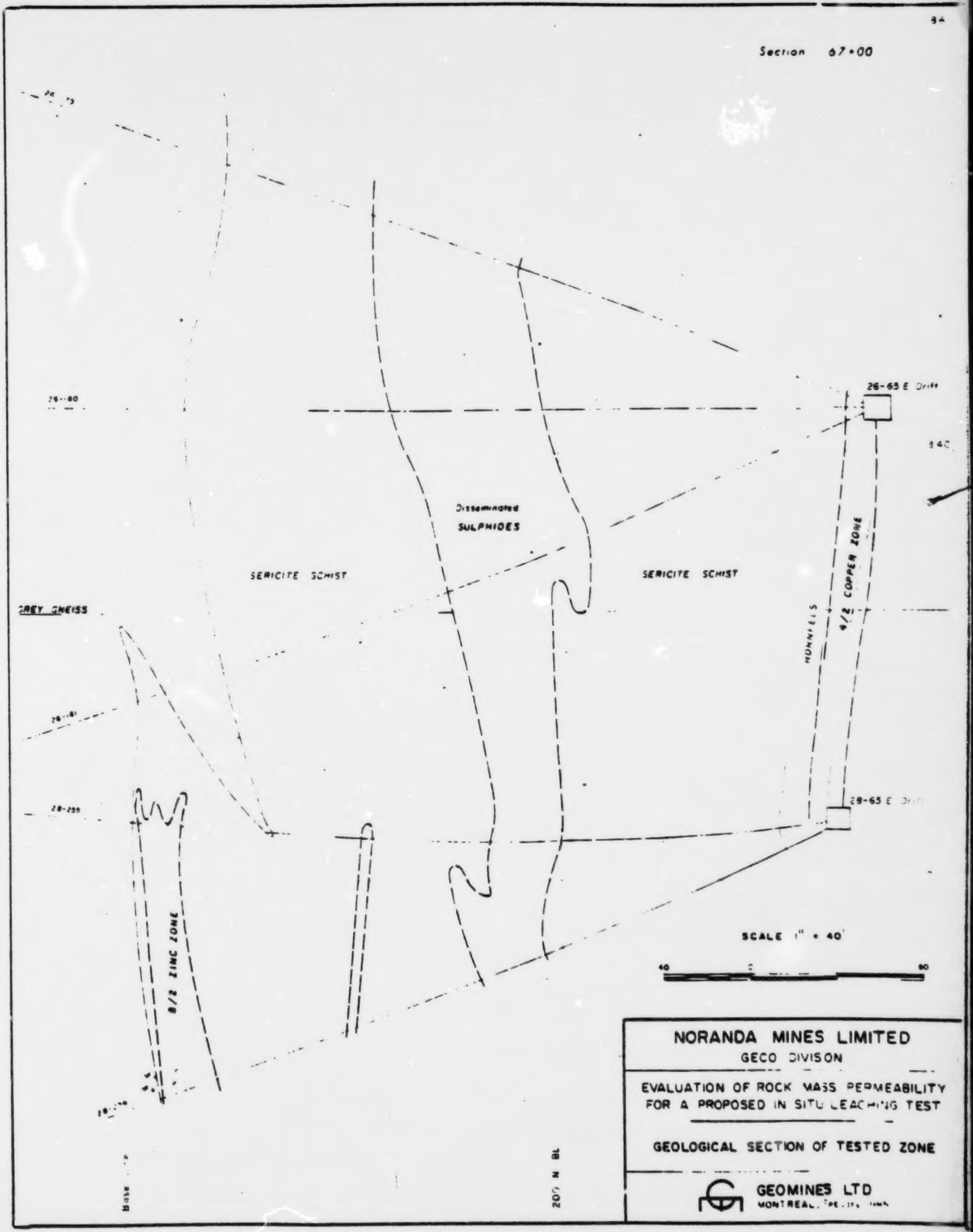
FAULT
Disseminated
SULPHIDES
X-CUT
SCALE 1" = 20'
20'
0
40'

NORANDA MINES LIMITED
GECO DIVISION

EVALUATION OF ROCK MASS PERMEABILITY
FOR A PROPOSED IN SITU LEACHING TEST

PLAN VIEW OF TEST ZG
WITH DRILL HOLE LOCATION

GEOMINES LTD
MONTREAL, February 1983

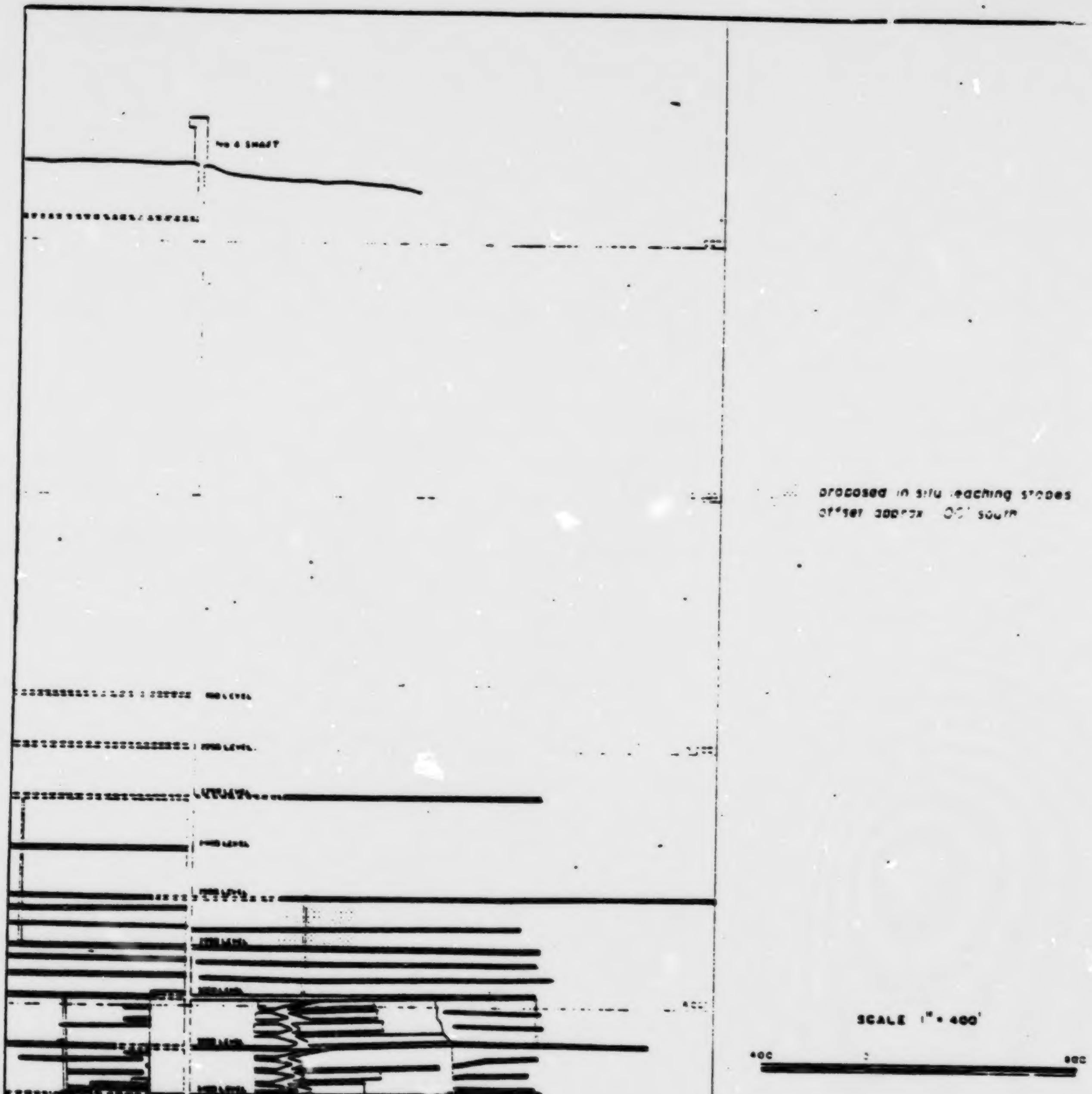

Section 67+00
26-80
26-181
28-239
GREY GNEISS
SERICITE SCHIST
Disseminated
SULPHIDES
SERICITE SCHIST
26-65 E Drift
340
HORNFELS
4/2 COPPER ZONE
28-65 E Drift
8/2 ZINC ZONE
Base
200 N BL
SCALE 1" = 40'
40
0
80

NORANDA MINES LIMITED
GECO DIVISON

EVALUATION OF ROCK MASS PERMEABILITY
FOR A PROPOSED IN SITU LEACHING TEST

GEOLOGICAL SECTION OF TESTED ZONE

GEOMINES LTD
MONTREAL, [illegible]



no 4 SHAFT
proposed in situ leaching stopes
offset approx. 100' south
SCALE 1" = 400'
400
0
800

NORANDA MINES LIMITED
GECO DIVISION

EVALUATION OF ROCK MASS PERMEABILITY
FOR A PROPOSED IN SITU LEACHING TEST

MINE LAYOUT, EAST-WEST SECTION
VICINITY OF no 4 SHAFT

GEOMINES LTD
MONTREAL, February 1982

APPENDIX B

MINING TERMS AND SELECTION CRITERIA

MINING TERMS:

-Alimak Raise: (See Raise)	A raise driven using a retractable platform that runs on rails attached to the host rock.
-Back:	The roof or overhead surface of an underground excavation.
-Benching:	A sequential method of blasting.
-Bulkhead:	A barrier constructed in an underground opening to restrict access or the flow of material.
-Burden:	The distance between consecutive rows of drill holes.
-Crosscut:	A nearly horizontal or horizontal underground drift that is driven to intersect an orebody and is normally used for access and then subsequently for drilling and blasting.
-Charge:	The density (w/w) of explosive used to fragment rock.
-Dragfold:	Used to define adjacent anticlinal (concave) and synclinal (convex) folds on the limb of a larger fold.
-Drawpoints:	A place where ore can be loaded and subsequently, removed from the bottom of a stope.
-Drift:	Horizontal or nearly horizontal underground opening typically used as a roadway.
-Muck:	Broken ore or waste rock.
-Mucking:	The removal of muck from a stope or development drift.
-Overcut:	An opening made on the top of a stope for drill set-up.
-Raise:	Underground opening that is driven upwards from one level to another by one of a variety of methods.
-Rockbolts:	Long, self-anchoring bolts used to stabilize the back and walls of underground openings.

-Slash:	Increase the dimensions of an underground opening with further drilling and blasting.
-Slot:	Vertical or inclined ore section that is excavated in a stope to allow for further blasting and the resulting ore swell.
-Slot Raise:	A raise driven where the slot is to be opened up to allow for blasting of the slot.
-Spacing:	The distance between the toes of two drill holes.
-Stope:	An underground excavation that is made by removing ore from the surrounding rock.
-Sub-level:	A system of secondary horizontal openings situated between main levels developed for drilling and blasting of stopes.
-Synform:	A term used to describe a syncline when the stratigraphical succession is unknown.

MINING SELECTION CRITERIA

Selecting a mining method depends on both technical and non-technical considerations, e.g.,

TECHNICAL	NON-TECHNICAL
Rock strength and competence, geology (location, dip, structures), grade, tonnage.	Available personnel, safety, future innovations.

At the Geco Mine, a blasthole or sub-level stoping technique was selected as the primary mining method in the 1950's on the basis of the above parameters. To use this type of method, the following characteristics are needed:

- -steeply dipping orebody,
- -competent host rock and orebody,
- -regular ore contacts.

The decision to use a modified form of the blasthole technique in place at Geco for the purpose of this investigation was based on, in part, the fact that the zone investigated has the required blasthole characteristics and that all the equipment, manpower, and expertise needed for this type of mining, already exist at Geco, thereby making the project less capital intensive. Any modifications (e.g., drilling and blasting) were used for either technical or cost savings reasons.

Volume

3

CENTRE DE RECHERCHE NORANDA

CONTRACT REPORT NO.0SQ84-00231

CONTRACT RESEARCH REPORT
N-8515 C.R. #85-02

A PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT

DSS FILE NO. 07SQ. 23440-4-9071

MILESTONE 3 REPORT

PRE-FEASIBILITY OF THE METALLURGICAL CIRCUIT

Project N-8515

March 1985

DECLASSIFIED
DATE 18/2/86
AUTHORIZED BY

115pp

Department of Extractive Metallurgy
Centre de Recherche Noranda
240 boulevard Hymus
Pointe Claire, Québec
H9R 1G5

MICROMEDIA

PART I

a) Presently, the project is behind schedule. The reason for the delay was explained during our meeting on December 17, 1984 and was stated in the Milestone 1 report.

2) The project is within budget.

3) The project is free of any areas of concern on which Crown assistance or guidance is required.

4) Milestone 4 report will follow shortly.

PART II

a) Phase 3, the Pre-Feasibility of the Metallurgical Circuit, has been completed as required. Work was started on Phase 4. The whole project will be completed by the end of March, 1985.

b) No variation from the plan of work was made during the completion of Phase 3.

c) No major items of equipment were purchased during the completion of Phase 3.

d) No trips were made during this phase.

PART III

A copy of the "Plan of Work and Cost" follows this page. It illustrates the present time allowance for each task and the work completed as of March 28, 1985.

TABLE A

Supply and Services Canada Approvisionnements et Services Canada

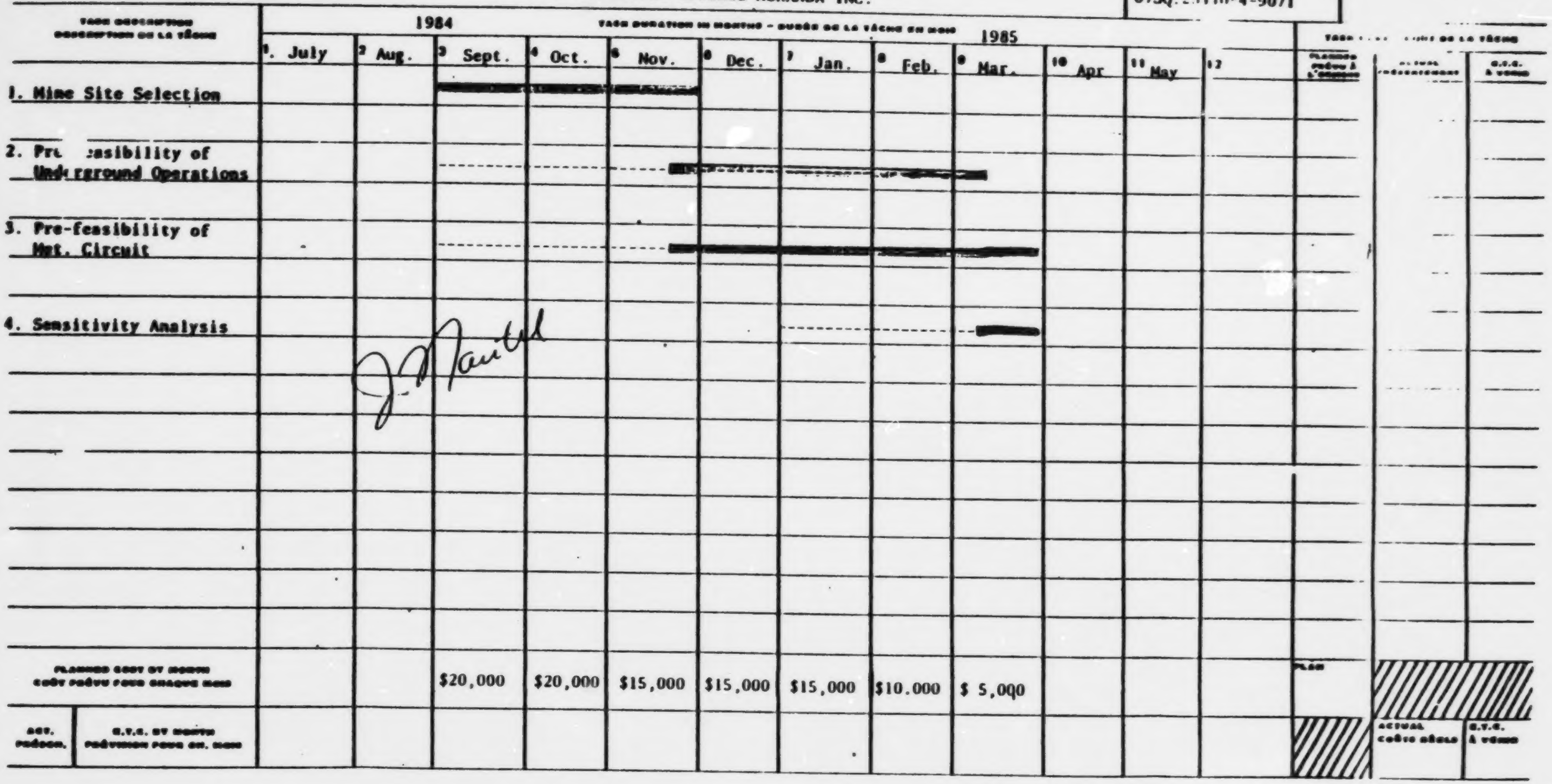
TIME/LOST CHART - *

CONTRACT PLAN AND REPORT FORM – FORMULE DE PROJET DE CONTRAT ET DE RAPPORT

CONTRACT TITLE – TITRE DU CONTRAT 'PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT'

CONTRACTOR – ENTREPRENEUR Centre de Recherche Noranda-NORANDA INC.

FILE NO. - Nº DU DOSSIER 07SQ.23110-4-9071



TASK DESCRIPTION DESCRIPTION DE LA TÂCHE	1984 1. July	2 Aug.	3 Sept.	4 Oct.	5 Nov.	6 Dec.	1985 7 Jan.	8 Feb.	9 Mar.	10 Apr	11 May	12	TASK COST - COÛT DE LA TÂCHE PLANNED PRÉVU À L'ORIGINE	ACTUAL	E.T.C. À VENIR
1. Mine Site Selection															
2. Pre-feasibility of Underground Operations															
3. Pre-feasibility of Met. Circuit															
4. Sensitivity Analysis															
PLANNED COST BY MONTH COÛT PRÉVU POUR CHAQUE MOIS			$20,000	$20,000	$15,000	$15,000	$15,000	$10.000	$ 5,000				PLAN		
ACT. PRÉVU. E.T.C. BY MONTH PRÉVISION POUR CH. MOIS														ACTUAL COÛTS RÉELS	E.T.C. À VENIR

J. Martel

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(en français)

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EXECUTIVE SUMMARY

In Phase III of this project, a metallurgical circuit for the production of 1000 tonnes per year copper by in-situ bacterial leaching of an underground sulphide deposit containing 0.9% Cu as chalcopyrite followed by copper recovery by cementation and solvent extraction-electrowinning was designed and costed. The design parameters were based upon a 55% copper recovery over a 10-year period from an average 15 cm size particle.

The initial capital investments of the leaching and cementation circuits were estimated at $1.6 million and $1.7 million, respectively, for a total initial investment of $3.3 million. The operating costs were estimated to be $1.14/kg Cu (51.5¢/lb). Smelting, refining and transportation charges result in total production costs of $1.47/kg Cu (66.8¢/lb).

A solvent extraction-electrowinning alternative requies a total capital investment of $6.4 million and operating costs of $1.12/kg Cu (50.8¢/lb). Transportation of cathodes to market would increase the total production costs to $1.21/kg Cu (54.9¢/lb).

On the basis of better economics and technical adaptability to in-situ leaching, the cementation recovery route was selected as the best alternative for this project

SOMMAIRE EXECUTIF

Durant la phase III de ce projet, un circuit métallurgique pour une production annuelle de 1000 tonnes de cuivre fut conçu et estimé. Ce circuit consiste à faire en phase, la lixivation bactériellement assistée sur un dépôt de sulfure de cuivre à base teneur (0.9% Cu) sous forme de chalcopyrite, suivie par une vécupération du cuivre par cémentation ou par extraction par solvant-électrolyse. Les critèris de conception furent basis sur une récupération de 55% en dix ans àpartir de particules de minerai dont la dimension moyenne est de 150 mm.

Le investissements initiaux pour les circuits de lixiviation et de cémentation furent estimés à 1,6 M$ et 1,7 M$ respectivement, pour un total de 3,3 M$. Les coûts d'opération furent estimés à 1,14 $/kg de cuivre (51.5¢/lb). Les frais de smeltage, d'affinage et de transport portent les coûts de production à 1,47$/kg (66,84 ¢/lb).

L'extraction par solvant-électrolyse d'un autre côté requiert un investissement total de 6,4 M$ et de coûts d'opération de 1,12 $/kg (50,8 ¢/lb). Les frais de transport des cathodes portent le coût total de production à 1,21 $/kg (54,9 ¢/lb).

Pour des raisons économiques et techniques, la récupération par voie de cémentation est recommandée pour un circuit de cette envergure.

SUMMARY

1. The design of the metallurgical circuit for the recovery of 1000 tonnes copper/year by in-situ bacterial leaching of a 0.9% Cu (all sulphide) ore, 1000 m below surface, consists of:

    a. Sprinkling ferric sulphate solution innoculated with bacteria on top of rubblized ore contained in stopes 31 m long x 9 m wide x 61 m deep, at an application rate of 10 $L/m^2/h$.

    b. Recoverying pregnant leach solution at the draw points from under the broken stope and either recirculating it to the top of other stopes or pumping it to surface if the concentration is at 1 gpL Cu.

    c. Precipitating copper in an iron cementation plant as 85% cement copper which is filtered and sent to a smelter, and returning the barren solution underground after adjustment to 0.3 gpL H_2SO_4.

2. The capital costs of the leaching and cementation circuits are $1.6 million and $1.7 million, respectively, for a total initial investment of $3.3 million (Cdn). These costs are exempt from mining costs.

3. The total operating costs for producing cement copper are 51.5¢/lb Cu. Smelting, refining and transportation charges result in a total refined metal cost of 66.8¢/lb Cu.

4. An SX-EW plant alternative would require a capital cost of $6.4 million for the whole project and would have an operating cost of 50.8¢/lb Cu. Additional transportation costs should be added to the saleable cathode copper, resulting in a total production cost of 54.9¢/lb Cu.

5. The selection of the cementation route was believed to be the best alternative for the solution management problem of this project. During the beginning and final years of operation, the recovery plant will receive smaller volumes of pregnant leach solution than the design value, since stopes will be commissioned or taken off production. Cementation launder modifications are expected to be simpler than the SX mixer-settler system.

6. A cementation plant is more adaptable than a SX-EW plant to variable copper production as will be experienced during the first 4 years and last 2-6 years of the project. It has less unit operations and requires less supervision and as a result can be run with less operators at periods of low production.

7. The heat balance indicates that, under the set of assumptions taken for this study, the leach temperature will be between 25 and 45°C, and solution heating would not be required.

8. A major cost saving could result if the cementation plant were built underground. 87% of the major equipment costs in the leaching section and 66% of the leaching operating costs are incurred in the pumping of a 1 gpL pregnant leach solution to surface. This technology, however, has never been used.

9. The principal process criteria used in this study are:

Leaching

- Total copper extraction = 55%
- Leaching time/stope = 10 years
- Cu grade = 0.9%
- Solution application rate = 10 $L/m^2/h$
- Void space = 25% of stope volume
- % void occupied by sol'n = 25%
- Solution flow = constant (no rest periods)
- Initial rock temperature = 15°C
- Permeability = constant throughout the project life

- Iron extraction
  - from chalcopyrite = 55%
  - from pyrrhotite = 100%
  - from pyrite = 55%
- Oxygen availability = from solution and void air replacement
- Solution heating = not required
- Temperature of leaching = 25-45°C

Recovery

- Production = 1000 tonnes/year
- Pregnant leach solution concentration = 1 gpL Cu, 4 gpL Fe
- Flow = 114 m^3/h at full production
  = <114 m^3/h first 4 and last 6 years
- Pregnant temperature = 22°C
- Iron consumption in cementation = 2.5 kg/kg copper
- Organic losses in SX = 100 ppm
- Grade of cement copper = 85% Cu

1. INTRODUCTION

This report presents the design and cost estimates of the metallurgical circuit which would be required for producing 1000 tonnes/year of copper by in-situ bacterial leaching of a low-grade (0.9% Cu) sulphide zone, 1000 m below surface and using some of the services of an operating mine.

In Phase I it was decided that the Geco mine in Manitouwadge, Ontario, was the best of the Noranda mines evaluated, on which a pre-feasibility study of in-situ bacterial leaching could be carried out. Low-grade ore at Geco has been estimated at 2.5 million tonnes at a grade of 0.5% Cu.

The mining development required prior to leaching was studied in Phase II and it was decided that due to the proximity of the leachable zone to the operating mine and the characteristics of the orebody, leaching should be done by percolating the solution through rubblized ore within stopes that measure 31 m long x 9 m wide x 61 m deep. In so doing, about 25% of the stope volume is removed and taken to surface by conventional mining to allow for swelling during blasting, and about 25% of the orebody remains unbroken in pillars and roofs to maintain mine stability.

With the values mentioned and a recovery rate of 55% over 10 years, this project would produce only 390 tonnes copper/year. This value was considered uneconomic for an investment in a new recovery plant and to complete the feasibility study it was decided to assume that the orebody contained sufficient copper to produce 1000 tonnes/year. All other data of Geco (mineralogy, rock temperature, ventilation and mine geometry) have been respected.

Flowsheets have been designed for two recovery systems: cementation and solvent extraction-electrowinning(SX-EW).

Material and heat balances were estimated based on the mineralogy of Geco ore and the characteristics of the recovery processes. There were no experimental data on this project and no metallurgical tests done with this ore. The design criteria for the prefeasibility study were decided on the basis of published information pertaining to (i) bacterial and in-situ work done by others at the laboratory and/or pilot plant scale and (ii) operating experience in related systems.

2. GENERAL DESCRIPTION OF IN-SITU BACTERIALLY-ASSISTED LEACHING

2.1 Ore Rubblization

The orebody is divided into sections measuring ∿40 m long, 15 m wide and 70 m deep. A stope of rubblized material is prepared within each of these,which will measure 31 m long, 9 m wide and 61 m deep. This task is the subject of phase II of this project and is covered in detail in the Milestone 2 report. Basically it consists in opening accesses from the existing shaft to the top and bottom levels of the future stopes, removing slots of ore from the lateral sides of the stope so that the remaining rock can swell when it is blasted, drilling and blasting the ore to rubblize it to an average maximum size of 15 cm.

2.2 Leaching

Leach solution, containing acidified iron sulphate and bacteria, (barren from the recovery plant) is brought from the surface down to the level at the top of the stope and transferred through pipes to sprinklers which are layed down on top of the rubblized ore in the stopes. The solution is sprayed on and percolates through the rubblized ore, leaching the copper from it, and exits at the bottom of the stope where it collects in sumps to be transferred to surface or recirculated to other stopes.

2.3 Recovery

Pregnant leach solution is pumped to surface and processed in the recovery plant to obtain a saleable copper product. Barren is adjusted to the desired acidity and iron content and sent underground for leaching.

3. LIMITATIONS OF REAL CASE STUDY

3.1 Copper Reserves at Geco Mines

The total available low-grade ore within the areas outlined in Phase II of this study has been estimated to be only 2.5×10^6 tonnes at a 0.5% Cu grade.

Since about 25% of the deposit is removed by conventional mining to allow for swelling during blasting and 25% remains in pillars and roofs between stopes to maintain mine stability, the total contained copper available for leaching is only 7,031 tonnes.

3.2 Copper Extraction

No testwork has been carried out with Geco for this study nor is there any commercial application of in-situ bacterially-assisted leaching. A maximum copper extraction of 55% over a 10-year period was therefore assumed, based upon published laboratory and pilot plant studies for chalcopyrite ores.

3.3 Recovery Plant Size

The current technology for copper recovery from dilute sulphate solutions is cementation or solvent extraction-electrowinning. At the 55% copper recovery over a 10-year period assumed for this study, only 390 tonnes Cu would be recovered per year. This limited copper production does not justify an investment in any new recovery plant.

4. ESTABLISHMENT OF BASE CASE CONDITIONS FOR FEASIBILITY STUDY

4.1 Production Tonnage

In order to complete the prefeasibility study, it was decided to assume a base case with a production of 1000 tonnes of copper per year. It is also assumed that the ore reserves required for this tonnage are available in the same orebody at Geco by increasing the reserves to 3.8×10^6 tonnes and improving the grade to 0.9% Cu. It was, therefore, possible to establish mine development costs based on actual data and use the mineralogy of this orebody in studying the chemistry and heat balances of the leaching process.

The production tonnage selected is a compromise. It is still small for SX-EW plants but only three times greater than estimated tonnages of low grade ore at Geco.

4.2 Pregnant Leach Solution

Based on available information from bacterial leaching studies and dump leach operations, the copper concentration in the pregnant leach solution pumped to surface was fixed at 1 gpL.

4.3 Recovery Plant Alternatives

Both cementation and solvent extraction-electrowinning processes were considered since an annual production of 1000 tonnes is smaller than commercial SX-EW plants. The recovery plant interacts with the leaching process by generating or consuming acid and by producing iron so that each alternative was studied in conjunction with the underground operation and solution treatment/residue impoundment alternative.

4.4 Process Flowsheet

The generalized flowsheet considered for this study for the in-situ bacterially-assisted leaching of low-grade chalcopyrite ore followed by copper recovery from the leach liquor by either cementation of solvent extraction-electrowinning is presented in Figure 1.

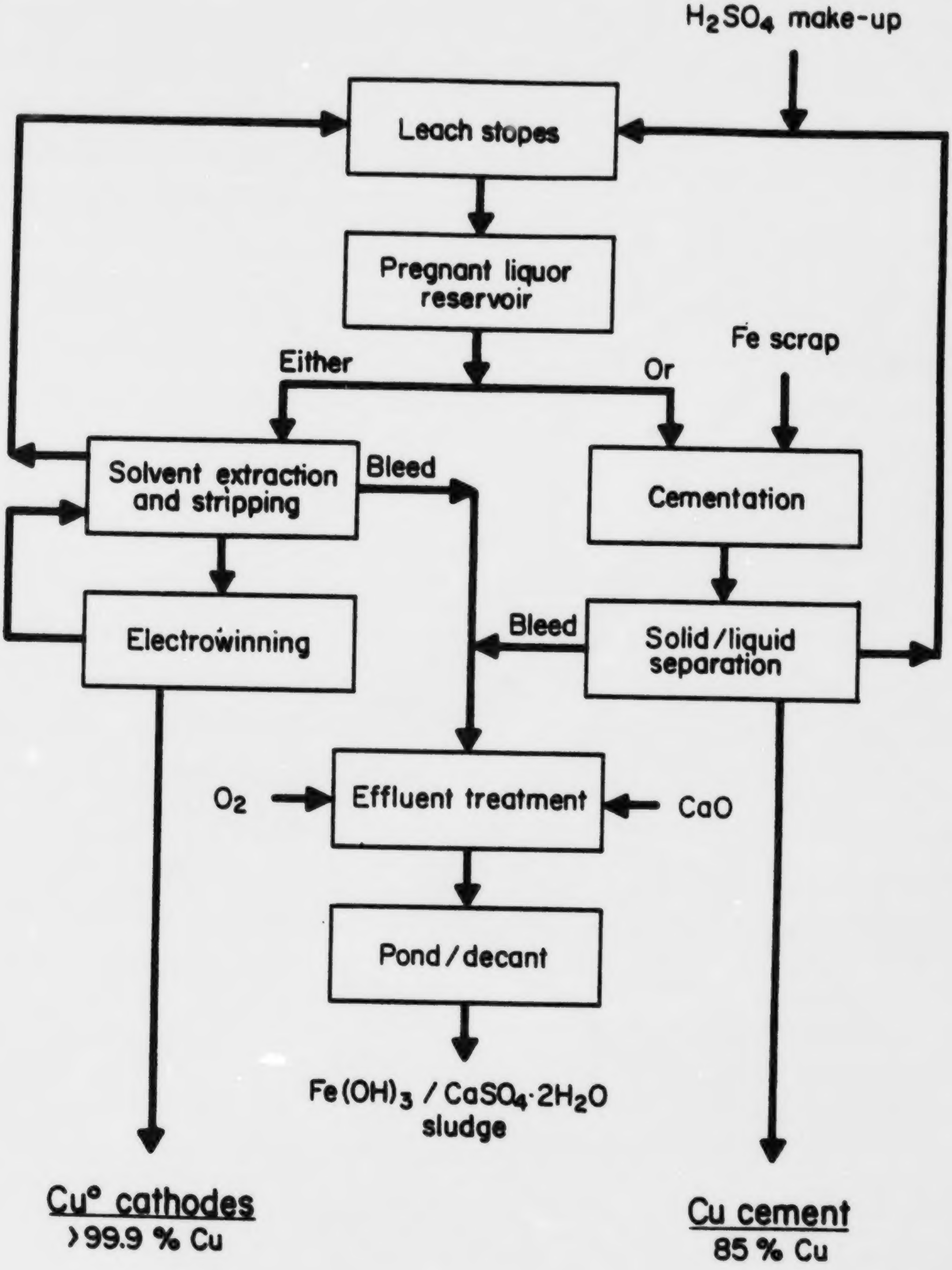
H_2SO_4 make-up
Leach stopes
Pregnant liquor reservoir
Either
Or
Fe scrap
Solvent extraction and stripping
Bleed
Cementation
Electrowinning
Bleed
Solid/liquid separation
O_2
Effluent treatment
CaO
Pond/decant
$Fe(OH)_3$ / $CaSO_4 \cdot 2H_2O$ sludge
Cu° cathodes
>99.9 % Cu
Cu cement
85 % Cu

Figure 1 - Generalized Flowsheet for In-Situ Leaching and Copper Recovery

5. LEACHING PROCESS DESIGN

5.1 Geco Ore Composition

The mineralogical and chemical composition of the rock contained within the boundaries outlined for this feasibility study are given in Tables I and II and were estimated from actual drill hole data. It is a zone of disseminated sulphides consisting of chalcopyrite, pyrrhotite and pyrite in a quartz host rock.

For the purpose of this study, it was assumed that all stopes have the same chemical and mineralogical composition.

TABLE I

MINERALOGICAL COMPOSITION OF GECO ORE CONSIDERED FOR STUDY

Mineral	Formula	Grade,%
Chalcopyrite	$CuFeS_2$	2.60
Pyrrhotite	FeS	2.86
Pyrite	FeS_2	1.49
Gangue	Quartz	93.05

TABLE II

CHEMICAL COMPOSITION OF GECO ORE

Element	%
Cu	0.9
Fe_T	3.2
S_T	2.8

5.2 Leaching Reactions

The leaching of sulphide minerals with an acidified ferric sulphate solution, innoculated with thiobacillus ferrooxidans bacteria may proceed by both direct and indirect bacterial catalysis. In the indirect mechanism ferric ions, which originate from pyrite and pyrrhotite dissolution, oxidize the sulphides and are in turn reduced to the ferrous state. The ferrous ions are then reoxidized to the ferric state with dissolved oxygen through bacterial catalysis. In the direct mechanism, dissolved oxygen acts directly on the mineral surface in the presence of bacteria. The principal reactions considered to occur during the acid-bacterial leaching of low-grade rock containing separate inclusions of chalcopyrite, pyrrhotite and pyrite are given in Table III, together with the estimated heats of reaction.*

TABLE III

HEATS OF REACTIONS CONSIDERED FOR BACTERIALLY-ASSISTED LEACHING OF SULPHIDE MINERALS

Reaction	Heat of Reaction kJ
$FeS_2 + 3.5\ O_2 + H_2O \xrightarrow{bact.} FeSO_4 + H_2SO_4$ (1)	-1443.19
$FeS_2 + Fe_2(SO_4)_3 \rightarrow 3FeSO_4 + 2S^\circ$ (2)	+8.039
$FeS + 2\ O_2 \rightarrow FeSO_4$ (3)	-903.93
$4FeSO_4 + O_2 + 2H_2O \rightarrow 2Fe_2(SO_4)_3 + 2H_2O$ (4)	-413.66
$CuFeS_2 + 2Fe_2(SO_4)_3 \rightarrow CuSO_4 + 5FeSO_4 + 2S^\circ$ (5)	+5.82
$2CuFeS_2 + 8.5O_2 + H_2SO_4 \rightarrow 2CuSO_4 + Fe_2(SO_4)_3 + H_2O$ (6)	-3511.05
$2\ S^\circ + 3\ O_2 + 2\ H_2O \rightarrow 2\ H_2SO_4$ (7)	-1244.32
$3\ Fe_2(SO_4)_3 + 10\ H_2O \rightarrow 2\ Fe_3(SO_4)_2(OH)_5 + 5H_2SO_4$ (8)	+333.69
$CaCO_3 + H_2SO_4 + H_2O \rightarrow CaSO_4 \cdot 2H_2O + CO_2$ (9)	-14.40

---

* For this study, it has been assumed that all sulphide minerals that react are oxidized completely to ferric sulphate, copper sulphate and sulphuric acid.

These reactions have been used for calculating the acid and iron balances, estimating the oxygen requirements (to establish if air injection is required) and predicting the solution temperature within the stopes.

5.3 Rates of Extraction

There are no metallurgical data available for this ore and therefore all extraction rate values were assumed based upon a review of published information on bacterial leaching in dumps and experimental laboratory and large column tests.

For the base case study, it was assumed that chalcopyrite leaches at an average rate of 0.015% per day to give a maximum recovery of 55% copper over a 10-year period. Pyrrhotite leaches much more readily than chalcopyrite and it was assumed that 100% dissolution will occur over 10 years. Pyrite oxidation was assumed to be the same as chalcopyrite at 55% recovery over 10 years.

The annual copper and iron extraction were estimated on a per stope basis from typical leaching rate profiles and are shown in Table IV.

5.4 Stope Development Schedule

It is assumed that (i) the total production of copper will be 10,000 tonnes, (ii) the target annual production will be 1000 tonnes per year and (iii) each stope will be leached for 10 years. At a grade of 0.9% Cu and 55% copper recovery, the minimum ore required is 3.8×10^6 tonnes. The total number of stopes required to meet this capacity, using the stope configuration outlined in Phase II, was estimated to be 57. For ease of calculation, however, it was assumed that 60 stopes would be developed (Table V).

TABLE IV

ESTIMATED RATES OF COPPER AND IRON DISSOLUTION PER STOPE[1]

Year	Chalcopyrite Copper Annual mt	Chalcopyrite Copper Cumulative mt	Chalcopyrite Iron[2] Annual mt	Chalcopyrite Iron[2] Cumulative mt	Pyrrhotite Iron[3] Annual mt	Pyrrhotite Iron[3] Cumulative mt	Pyrite Iron[4] Annual mt	Pyrite Iron[4] Cumulative mt	Total Iron Annual mt	Total Iron Cumulative mt
1	31.8	31.8	27.9	27.9	116.9	116.9	24.6	24.6	169.4	169.4
2	28.7	60.5	25.2	53.1	105.2	222.1	22.1	46.7	152.5	321.9
3	25.5	86.0	22.4	75.5	93.5	315.6	19.6	66.3	135.5	457.4
4	22.3	108.3	19.6	95.1	81.8	397.4	17.2	83.5	118.6	576.0
5	19.1	127.4	16.8	111.9	70.1	467.5	14.7	98.2	101.6	677.6
6	16.0	143.4	14.1	126.0	58.4	525.9	12.3	110.5	84.8	762.4
7	12.7	156.1	11.2	137.2	46.7	572.6	9.8	120.3	67.7	830.1
8	9.5	165.6	8.4	145.6	35.1	607.7	7.4	127.7	50.9	881.0
9	6.4	172.0	5.6	151.2	23.4	631.1	4.9	132.6	33.9	914.3
10	3.1	175.0	2.7	153.9	11.7	642.8	2.4	135.0	16.8	931.1

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[1] 35,356 mt rock per stope at 0.9% Cu, 3.3% Fe

[2] Total dissolution of chalcopyrite : 55%

[3] Total dissolution of pyrrhotite : 100%

[4] Total dissolution of pyrite : 55%

TABLE V

ESTIMATED STOPE REQUIREMENTS

STOPE DIMENSION	30.6m x 9.1m x 59.2m
VOLUME /STOPE	16,480 m^3
WEIGHT /STOPE	47,140 mt
ORE REMOVED FOR BLAST (25%)	11,785 mt
ORE AVAILABLE FOR LEACH/STOPE	35,355 mt
COPPER CONTENT (0.9%)	318 mt
MAXIMUM Cu RECOVERY/STOPE	175 mt
TOTAL COPPER PRODUCTION	10,000 mt
TOTAL STOPES REQUIRED	57.1 say 60
TOTAL RESERVES REQUIRED including 25% for pillars	3,771,200 mt
TOTAL LEACHABLE ORE	2,121,300 mt

Several schedules were evaluated for the development of the 60 stopes as shown in Tables VI to VIII. In order to achieve the target annual copper production of 1000 tons in the shortest time and not interfere with the hoisting capacity of Geco's No. 4 shaft, it was decided to adopt the schedule in Table VIII. With this alternative, the project will produce 1000 tonnes/year copper in the fourth year of operation and will require that approximately 215 tonnes/day of development ore be removed, which represents approximately 10% of the hoisting capacity at Geco. Since there are normal variations of 20% in this parameter, the selected schedule is compatible with present mine activities.

TABLE VI

COPPER PRODUCTION SCHEDULE

(Development of 4 stopes/year)

tonnes/year

Stopes \ year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25
4	127	115	102	89	76	64	51	38	26	12															
8		127	115	102	89	76	64	51	38	26	12														
12			127	115	102	89	76	64	51	38	26	12													
16				127	115	102	89	76	64	51	38	26	12												
20					127	115	102	89	76	64	51	38	26	12											
24						127	115	102	89	76	64	51	38	26	12										
28							127	115	102	89	76	64	51	38	26	12									
32								127	115	102	89	76	64	51	38	26	12								
36									127	115	102	89	76	64	51	38	26	12							
40										127	115	102	89	76	64	51	38	26	12						
44											127	115	102	89	76	64	51	38	26	12					
48												127	115	102	89	76	64	51	38	26	12				
52													127	115	102	89	76	64	51	38	26	12			
56														127	115	102	89	76	64	51	38	26	12		
60															127	115	102	89	76	64	51	38	26	12	
																127	115	102	89	76	64	51	38	26	12
Total Cu Production mt/y	127	242	344	433	509	573	624	662	688	700	700	700	700	700	700	700	573	458	350	267	191	127	76	38	12
Stopes in Operation	4	8	12	16	20	24	28	32	36	40	40	40	40	40	40	40	36	32	28	24	20	16	12	8	4

TABLE VII

COPPER PRODUCTION SCHEDULE

(Development of 6 Stopes/year)

tonnes/year

Stopes / Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
6	191	172	153	134	115	96	76	57	38	19									
12		191	172	153	134	115	96	76	57	38	19								
18			191	172	153	134	115	96	76	57	38	19							
24				191	172	153	134	115	96	76	57	38	19						
30					191	172	153	134	115	96	76	57	38	19					
36						191	172	153	134	115	96	76	57	38	19				
42							191	172	153	134	115	96	76	57	38	19			
48								191	172	153	134	115	96	76	57	38	19		
54									191	172	153	134	115	96	76	57	38	19	
57										95	86	76	67	57	48	38	28	19	9
Total Cu Prod. mt/y	191	363	516	650	765	861	937	994	1032	955	774	521	468	343	238	152	85	38	9
Stopes in Operation	6	12	18	24	30	36	42	48	54	57	51	45	39	33	27	21	15	9	3

TABLE VIII

COPPER PRODUCTION SCHEDULE

(Development of 10 Stopes/year)
tonnes/year

Stopes / Year	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18
10	318	287	255	223	191	160	127	95	64	32								
20		318	287	255	223	191	160	127	95	64	32							
30			318	287	255	223	191	160	127	95	64	32						
40				318	287	255	223	191	160	127	95	64	32					
45					-	159	144	128	111	96	80	64	48	32	16			
50							159	144	128	111	96	80	64	48	32	16		
55								159	144	128	111	96	80	64	48	32	16	
60									159	144	128	111	96	80	64	48	32	16
Total Cu Prod mt/y	318	605	860	1083	956	988	1004	1004	988	796	605	446	319	224	160	96	48	16
Stopes in Operation	10	20	30	40	40	45	50	55	60	60	50	40	30	20	20	15	10	5

5.5 Rate of Leach Solution Application

An application rate of 10.2 L/m²/h was adopted for this study. This value is typical of sprinkler systems for dump leaching.

The low application rate was adopted to obtain the highest copper concentrations in the pregnant leach solution (PLS), thus compensating for the slow rates of reaction, and to maintain as much void volume as possible so that there would exist a proper aeration.

Lower application rates also result in lower pumping costs. Depending on the PLS concentration, the solution is either transferred to other stopes or pumped to surface for copper recovery.

5.6 Pregnant Leach Solution

It is assumed that the average pregnant leach solution (PLS) composition will be 1 g/L Cu so that the design volume of solution feed to the recovery plant will be 114 m³/h, based upon an annual copper production of 1000 tonnes. The solution management schedule is shown in Table IX.

The PLS volume fixes the design capacity of the copper recovery plant.

5.7 Solution Flow Management

Since new stopes are to be commissioned for leaching in each year in the first years of the project (10 stopes/year in the first 4 years), and taken off leaching at the end of the project, and it was assumed that the solution application rate will remain constant during the lifetime of the project, there will be variable volumes of PLS to pump to surface and the PLS from certain stopes will have to be recirculated through others during the middle years of the project.

TABLE IX

SOLUTION MANAGEMENT SCHEDULE

Year	No. of Operating Stopes	Total Surface Area	Application Rate		Pregnant Solution[1]
		m^2	m^3/h	m^3/d	Cu, gpL
1	10	2,785	28.5	684	1.27
2	20	5,569	57.0	1,368	1.21
3	30	8,354	86.0	2,052	1.15
4	40	11,138	114.0	2,736	1.08
5	40	11,138	114.0	2,736	0.96
6	45	12,531	114.0	2,736	0.99
7	50	13,923	114.0	2,736	1.00
8	55	15,315	114.0	2,736	1.00
9	60	16,708	114.0	2,736	0.99
10	60	16,708	114.0	2,736	0.80
11	50	13,923	114.0	2,736	0.61
12	40	11,138	114.0	2,736	0.45
13	30	8,354	86.0	2,052	0.43
14	25	5,569	57.0	1,368	0.45
15	20	5,569	57.0	1,368	0.32
16	15	4,177	28.5	684	0.38
17	10	2,785	28.5	684	0.19
18	5	1.392	14.0	336	0.13

[1] From a 0 gpL Cu feed solution.

The first factor affects the selection of the recovery plant method. In this aspect it was felt that the cementation plant offers the best solution since it can be designed for the maximum flow and smaller volumes would only result in unnecessary longer residence times, while with solvent extraction, PLS in the first and last years of operation will have to be mixed with barren to maintain the aqueous flow close to the designed volume. This would cause a certain dilution equivalent to what would occur if larger application rates were used to leaching.

The second factor was controlled by assuming that a group of ten stopes leaches at the same rate,(as a block) and building PLS sumps at a collection point from the ten stopes, so that if the PLS copper concentration is low, it can be pumped from this sump to another group of 10 stopes. Thus each 10-stope group was provided with one sump and two pumps (one stand-by).

5.8 Permeability and Irrigation Rate

The void space or pore volume within each stope was assumed to be equivalent to the rock swell during blasting, which is 25%. A fully saturated stope would therefore hold 4120 m^3 of lixiviant.

At an application rate of 10.2 $L/m^2/h$, the vertical flow velocity will be 0.25 m/d and under good ventilation conditions, the pore volume should contain sufficient oxygen to maintain the reactions assumed in Table III.

In many dump leaching operations the solution flow is interrupted for certain periods to allow the ore to "rest". During this time air is restored to the void space and higher extraction rates are observed when solution is re-applied.

Since, in this case, the bacteria population is an important factor to maintain proper leaching rates, it was felt that rest periods may be counter-active and therefore, continuous solution application tion has been adopted. As a result the irrigation rate is the same value as the application rate.

It has also been assumed that because leach solution flow is maintained constant there will not be any decrease in stope permeability during the leaching period of the stope.

5.9 Acid Balance

The acid balances for both the leach-SX-EW and cementation alternatives have been calculated based upon the reactions considered in section 5.3 and the assumption that all dissolved iron is oxidized to the ferric state and sulphur from the reactions is oxidized to sulphuric acid.

It was also assumed that the overall leaching system will be self-buffering between pH 2-2.8 and thus create a stable environment for bacterial activity as well as chalcopyrite and ferrous iron oxidation. In so doing, it was necessary to assume that the ore contains acid-consuming gangue.

Based upon the chemistry of this system, sulphuric acid will be consumed by (i) the dissolution of chalcopyrite, (ii) oxidation of ferrous iron to ferric iron and (iii) dissolution of the acid-consuming gangue and, generated by (i) the oxidation of sulphur to sulphate and (i) the precipitation of ferric iron as basic iron sulphate. For the SX-EW alternative, acid is also generated by the extraction of copper to the organic phase. The acid balances for both recovery alternatives are given in Tables X and XI.

TABLE X

ACID BALANCE FOR LEACH - SX-EW

ACID GENERATED FROM PYRITE OXIDATION,	mt	7,100	
ACID GENERATED FROM IRON PRECIPITATION,	mt	75,730	
ACID GENERATED FROM SX,	mt	16,194	
TOTAL ACID GENERATED,	mt		99,024
ACID CONSUMED FROM PYRRHOTITE OXIDATION,	mt	33,817	
ACID CONSUMED FROM CHALCOPYRITE OXIDATION,	mt	8,100	
ACID REMOVED IN BLEED,	mt	871	
ACID CONSUMED BY GANGUE, (by difference)	mt	56,236	
TOTAL ACID CONSUMED,	mt		99,024

TABLE XI

ACID BALANCE FOR LEACH - CEMENTATION

ACID GENERATED FROM PYRITE OXIDATION,	mt	7,100	
ACID GENERATED FROM IRON PRECIPITATION,	mt	110,787	
TOTAL ACID GENERATED,	mt	117,787	
TOTAL ACID FEED,	mt	3,871	
TOTAL ACID INPUT,	mt		121,758
ACID CONSUMED FROM PYRRHOTITE OXIDATION,	mt	33,817	
ACID CONSUMED FROM CHALCOPYRITE OXIDATION,	mt	8,100	
ACID CONSUMED BY GANGUE (by difference)		79,841	
TOTAL ACID CONSUMED,	mt		121,758

5.10 Iron Balance

Within the pH limits 2-2.8, created by the self-buffering leaching system, the total maximum ferric iron concentration is approximately 3-4 gpL. It was assumed that all the iron is oxidized to the ferric state, so that the total iron concentration in the pregnant liquor also at ∿3-4 gpL. It was also assumed that under steady state conditions, the net iron precipitation is equivalent to the iron released by sulphide dissolution plus iron introduced from cementation if iron precipitation of copper is used. The iron balance for both cementation and solvent extraction are shown in Tables XII and XIII.

5.11 Oxygen Requirements

The total oxygen required to oxidize 55% of the contained chalcopyrite, 100% of the contained phrrhotite and 55% of the contained pyrite within the 60 stopes to copper sulphate, sulphuric acid and ferric sulphate was estimated to be 89,551 mt. An additional 3,544 mt O_2 are required to oxidize the iron introduced for the cementation alternative for copper recovery.

It was assumed that air circulation within the stopes, which contain 25% voids, if free-flowing and that no air injection system is required in addition to the normal mine ventilation. The oxygen balance for both the SX-EW and cementation option is presented in Tables XIV and XV.

5.12 Heat Balance

The average net amount of heat generation as calculated from the standard heats of reaction (Table III) and assuming that all of the iron is oxidized to the ferric state, is 11.7 PJ. The most significant source of heat generation is pyrrhotite oxidation which contributes over 58% of the total heat, while the main heat losses are

TABLE XII

IRON BALANCE FOR LEACH - CEMENTATION

IRON LEACHED FROM PYRITE,	mt	8,092	
IRON LEACHED FROM PYRRHOTITE,	mt	38,542	
IRON LEACHED FROM CHALCOPYRITE,	mt	9,232	
IRON ADDED AS SCRAP,	mt	24,744	
TOTAL IRON SOLUBILIZED,	mt		80,640
IRON REMOVAL IN BLEED,	mt	4,759	
IRON PRECIPITATED IN STOPES,	mt	75,881	
TOTAL IRON REMOVAL,	mt		80,640

TABLE XIII

IRON BALANCE FOR LEACH - SX-EW

IRON LEACHED FROM PYRITE,	mt	8,092	
IRON LEACHED FROM PYRRHOTITE,	mt	38,542	
IRON LEACHED FROM CHALCOPYRITE,	mt	9,232	
TOTAL IRON SOLUBILIZED,	mt		55,866
IRON REMOVED FROM BLEED,	mt	3,996	
IRON PRECIPITATED IN STOPES,	mt	51,866	
TOTAL IRON REMOVAL,	mt		55,866

TABLE XIV

OXYGEN BALANCE FOR LEACH - SX/EW

O_2 REQUIRED FOR CHALCOPYRITE OXIDATION,	mt	22,474	
O_2 REQUIRED FOR PYRITE OXIDATION,	mt	17,387	
O_2 REQUIRED FOR PYRRHOTITE OXIDATION,	mt	49,640	
TOTAL OXYGEN REQUIREMENTS,	mt		89,551
AIR REQUIREMENTS [1]	m^3/s	3.0	

TABLE XV

OXYGEN BALANCE FOR LEACH - CEMENTATION

O_2 REQUIRED FOR CHALCOPYRITE OXIDATION,	mt	22,474	
O_2 REQUIRED FOR PYRITE OXIDATION,	mt	17,387	
O_2 REQUIRED FOR PYRRHOTITE OXIDATION,	mt	49,460	
O_2 REQUIRED FOR IRON FROM CEMENTATION,	mt	3,549	
TOTAL OXYGEN REQUIREMENTS,	mt		93,100
AIR REQUIREMENTS [1]	m^3/s	3.1	

[1] Assuming 20% oxygen efficiency. This is a guestimate since no relevant data is available.

(i) rubblized ore in the stope, (ii) pregnant leach solution which is pumped to surface and (iii) the air circulating through voids in the rubblized ore which then enters the main ventilation system of the mine.

For both recovery alternatives it was estimated that the heat losses in the transport of barren solution to the sprinklers and pregnant solution from the underground sumps to the recovery plant, most of which is done through pipes (2000 m) in the No. 4 shaft, result in a temperature drop of 5-7°C. The initial rock temperature at Geco is 15°C.

The heat balances for both recovery alternatives (Tables XVI and XVII) indicate that there will be temperature variations within each stope of between 15 and 45°C and that the average temperature of the leach solution exiting from the leached stopes could be between 27 and 29°C. The temperature of the PLS entering the recovery plant will average 22°C.

These calculations indicate that the underground bacterial leaching system and the recovery plant can operate without external solution heating.

5.13 Water Balance

It was assumed that there would be no solution losses within the underground leaching system and that the only bleed would be through the cement copper (15-20% moisture) in which case an equivalent amount of water will be added in the acidification stage.

Commencing in year 12 onwards, the volume of barren solution recycled to leaching will be decreased to maintain solution application rates at 10.2 $L/m^2/h$ and will be directed towards lime neutralization for iron and gypsum precipitation.

TABLE XVI

HEAT BALANCE FOR LEACH - SX-EW

ΔH_r FOR CHALCOPYRITE	-0.290 PJ
ΔH_r FOR PYRRHOTITE	-0.695 PJ
ΔH_r FOR PYRITE	-0.224 PJ
ΔH_r FOR BASIC IRON SULPHATE PRECIPITATION	+0.052 PJ
TOTAL HEAT GENERATED	-1.158 PJ
HEAT ABSORBED BY LEACHANT	0.054 PJ x ΔT
HEAT ABSORBED BY STOPES	1.172 PJ x ΔT
HEAT ABSORBED BY AIR	1.800 PJ x ΔT
TEMPERATURE RANGE °C	16-45

TABLE XVII

HEAT BALANCE FOR LEACH - CEMENTATION

ΔH_r FOR CHALCOPYRITE	-0.290 PJ
ΔH_r FOR PYRRHOTITE	-0.695 PJ
ΔH_r FOR PYRITE	-0.224 PJ
ΔH_r FOR CEMENT IRON OXIDATION	-0.046 PJ
ΔH_r FOR BASIC IRON SULPHATE ppt	+0.076 PJ
	-1.180 PJ
HEAT ABSORBED BY LEACHANT	0.054 PJ x ΔT
HEAT ABSORBED BY STOPES	1.172 PJ x ΔT
HEAT ABSORBED BY AIR	1.800 PJ x ΔT
TEMPERATURE RANGE °C	16-44

5.14 Description of Leaching System

Barren solution is transferred through a 15-cm diameter rubber-lined carbon steel (RLCS) pipe from the recovery plant, down the No.4 shaft to the 2650 level where it enters a 10-cm diameter pipe layed in the access drift to the orebody development. Solution is distributed through header pipes to each stope development access and transferred to 25-mm diameter pipes layed in the center of each rubblized stope. Five Senninger No.6 wobblers are joined to the distribution line at 6-m intervals. These sprinklers are impulse type. and discharge solution continuously over a 10-m diameter circle.

Solution emerging from the draw points under each stope are collected in drain ditches that convey it to a collection sump. Since in the stope development schedule it was estimated that the best commissioning option was 10 stopes/year, the solution emerging from each of these should have the same concentration of copper. Collection samples from each group of 10 or 5 stopes are connected to pump sumps. Each of these is provided with two 15-hp pumps to either transfer the PLS to the sprinkler system on another group of 10 stopes or to a common sump from where solution is pumped to the surface.

The pumping of 114 m^3/h of PLS from the 2850 level to the plant on surface is done through 15-cm diameter RLCS tubing in three stages of pumping, each of which is provided with two 150 hp pumps (one stand-by).

A schematic drawing of the underground piping system is presented in Figure 2.

Pressure let-down valves will be installed every 70 m between the surface and the distribution lines in the stopes in order to reduce pressure from ∿800 psi down to the 50-100 psi required in the sprinkler headers.

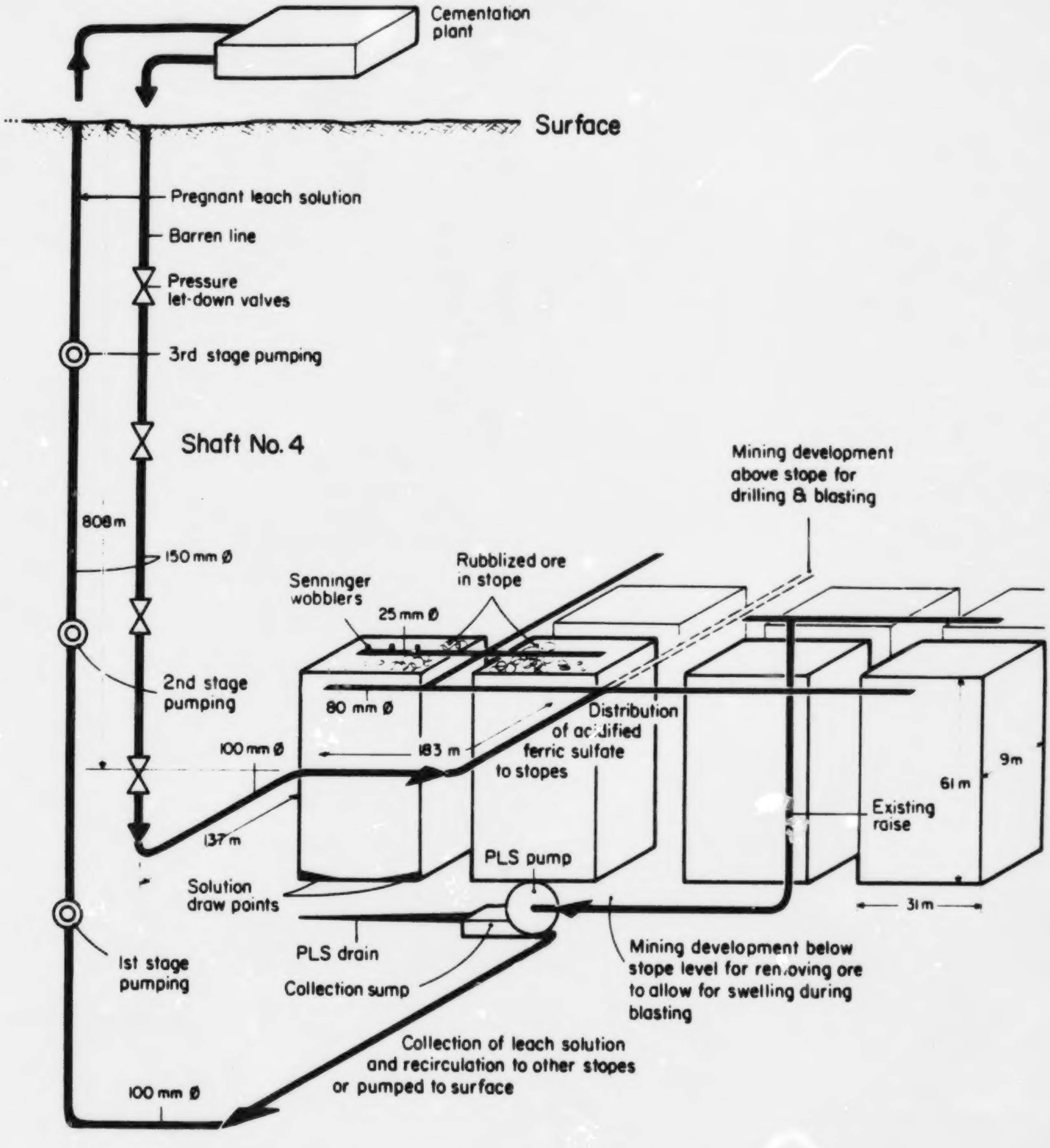
Cementation plant
Surface
Pregnant leach solution
Barren line
Pressure let-down valves
3rd stage pumping
Shaft No. 4
808m
150 mm Ø
2nd stage pumping
100 mm Ø
137 m
Senninger wobblers
25 mm Ø
Rubblized ore in stope
80 mm Ø
183 m
Distribution of acidified ferric sulfate to stopes
Mining development above stope for drilling & blasting
9m
61 m
Existing raise
31m
Solution draw points
PLS pump
PLS drain
Collection sump
1st stage pumping
Mining development below stope level for removing ore to allow for swelling during blasting
Collection of leach solution and recirculation to other stopes or pumped to surface
100 mm Ø

Figure 2. Schematic piping network for in-situ bacterial leaching

5.15 Labour Requirements

Due to the small scale of the recovery plant and the low labour requirements in the leaching system, it will be necessary that the operators perform multiple assignments. The main areas, in addition to the recovery plant, are:

leach field operations
laboratory
control Room
maintenance

It has been assumed that for the base case, at 1000 tonnes/year copper, the administration and engineering activities required for this project will be handled by the mine-mill personnel. In addition to the above activities, the project will have one single supervisor in charge of all project activities.

Total labour requirements for the leaching section for the base case study have been estimated as one person, including operation and maintenance.

The main activities in the leach system are:

pipe layout for sprinklers
pump maintenance
sampling of stope effluents

Some of these obviously require a two-person team, but manning of these activities will require personnel from other areas. It was assumed that no security personnel are required for this size of plant.

6. RECOVERY PLANT DESIGN

6.1 Plant Size

Both recovery alternatives were designed for treating 114 m^3/h of PLS solution at a 1 gpL copper concentration.

6.2 Cementation

The pregnant leach solution is fed to a series of launders containing detinned cans and copper is cemented out from the solution according to the following reaction

$$CuSO_4 + Fe \rightarrow FeSO_4 + Cu$$

at an iron consumption of 2.5 kg per kg copper precipitated. The cement passes through the false bottom of the launders to a settling tank. The barren solution overflow containing 0.03 gpL Cu, 5-6 gpL Fe, at pH 3 is acidified with sulphuric acid to 0.3 gpL H_2SO_4 and recycled to the stopes for further leaching. The cement copper is reslurried with water filtered on a plate and frame filter press and the cement cake which contains 15-20% moisture is transported to a smelter. A cement copper grade of 85% has been assumed.

The cementation plant flowsheet and equipment diagram is presented in Figure 3.

6.3 Solvent Extraction - Electrowinning

Copper is extracted from the pregnant leach solution by two stages of extraction with 5% LIX 64N in kerosene and stripped from the organic phase in one stage with spent electrolyte containing 170-180 g/L H_2SO_4 and 30 gpL Cu. The raffinate contains <0.01 gpL Cu, 4 gpL Fe and $\sim$1.5 gpL H_2SO_4 and is recycled to the stopes directly without pH adjustment.

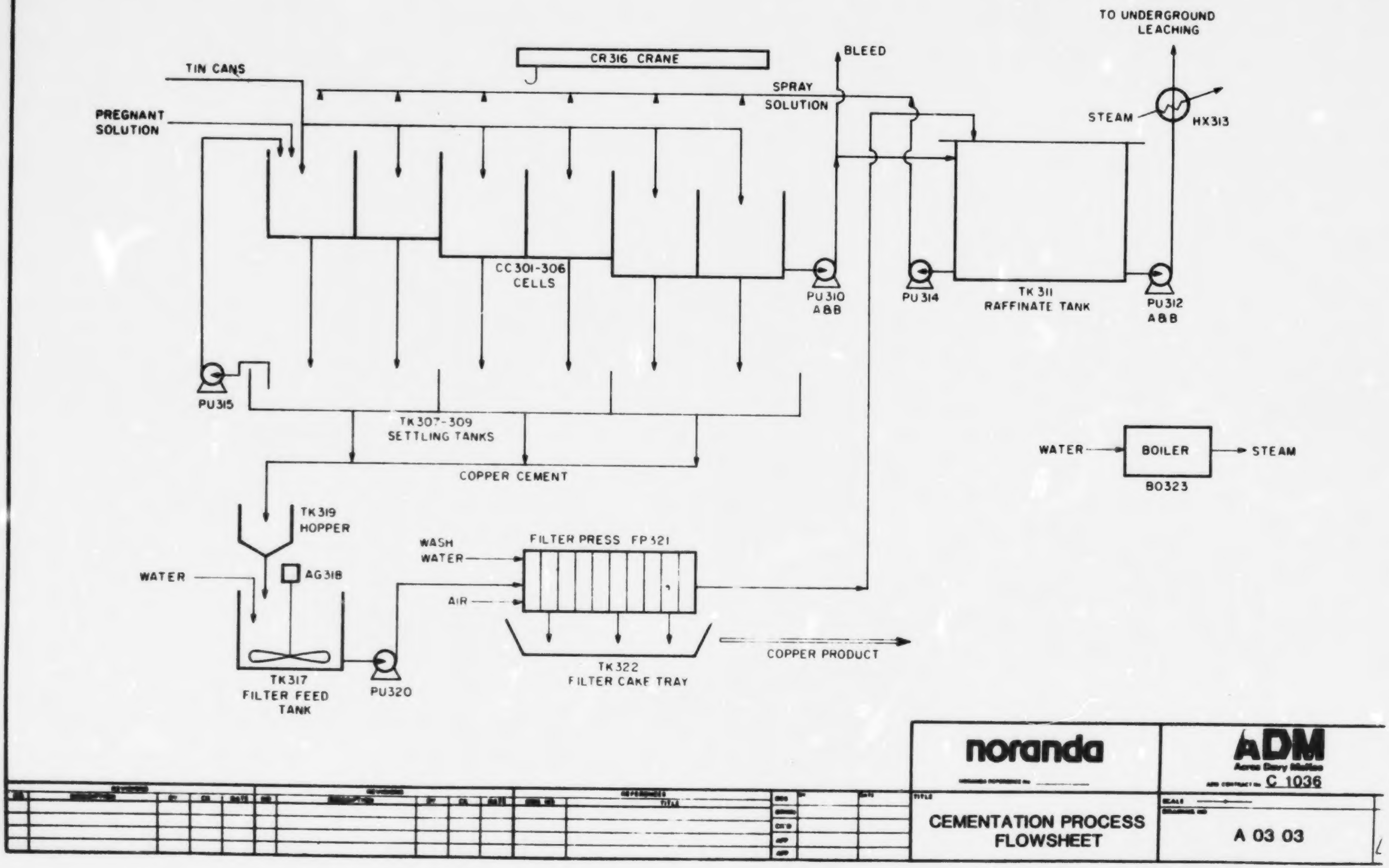
CR316 CRANE
BLEED
TO UNDERGROUND LEACHING
TIN CANS
PREGNANT SOLUTION
SPRAY SOLUTION
STEAM
HX313
CC301-306 CELLS
PU310 A&B
PU314
TK 311 RAFFINATE TANK
PU312 A&B
PU315
TK307-309 SETTLING TANKS
COPPER CEMENT
WATER
BOILER
STEAM
BO323
TK319 HOPPER
WASH WATER
FILTER PRESS FP321
WATER
AG318
AIR
COPPER PRODUCT
TK317 FILTER FEED TANK
PU320
TK322 FILTER CAKE TRAY
noranda
ADM
Acres Davy McKee
C 1036
CEMENTATION PROCESS FLOWSHEET
A 03 03

Figure 3 - Cementation Plant Flowsheet and Equipment Diagram

The pregnant electrolyte containing 45 gpL Cu and 150-160 gpL H_2SO_4 is fed to 14 commercial electrowinning cells that use lead/calcium/tin anodes and copper is electrowon at a current density of 215 A/m^2 and current efficiency of 85%. It was assumed that the quality of the copper cathodes is >99.9% and that they can be directly sold on the LME.

The SX-EW flowsheet and equipment diagram is presented in Figure 4 and a plant layout for both solvent extraction and electrowinning is shown in Figure 5.

6.4 Advantages of Cementation Over SX-EW

As mentioned in section 5.8, the recovery plant will either have to treat smaller PLS volumes than the designed capacity, during the first 4 years and the last 2-6 years of operation, or mix barren solution with PLS in order to pass a volume of aqueous solution to solvent extraction that is within a working range from the design value.

On this matter it seems obvious that a cementation plant offers an advantage over SX-EW plants because the designed residence time can easily be achieved by by-passing a certain amount of launders and because the major reagent consumption (iron) is a function of the copper tonnage recovered and less dependent on the volume of solution treated, whereas in solvent extraction the aqueous flow must be similar to the designed volume (maximum) and the major consumption (organic losses) is a function of the volume of solution treated.

At relatively comparable costs the cementation plant should be selected on the basis that it adapts better to an in-situ leaching operation.

Also, as will be discussed in section 8.7 on dealing with the final years of the project, operations in the recovery plant will have

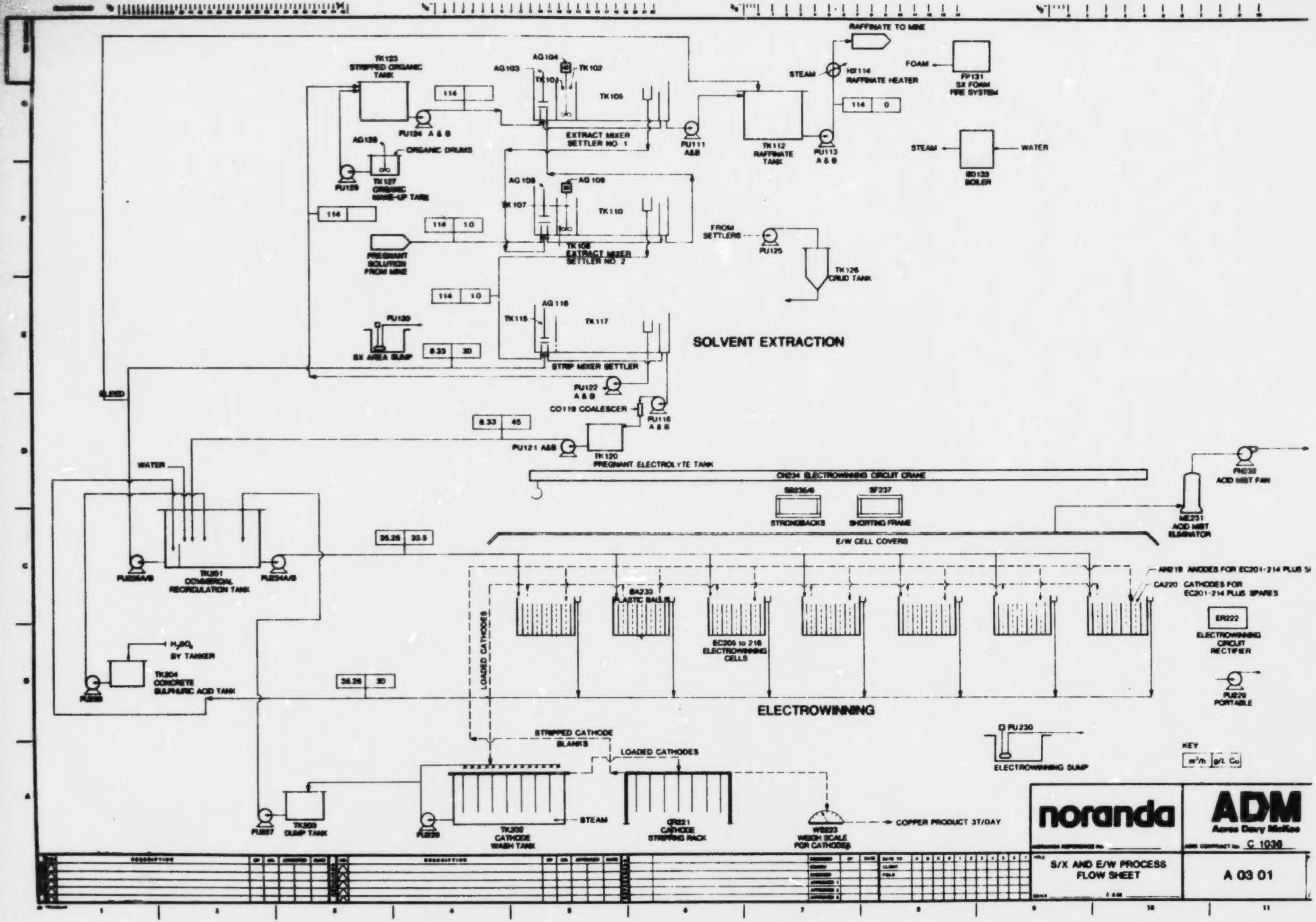
TK123 STRIPPED ORGANIC TANK
PU124 A & B
AG128
ORGANIC DRUMS
PU129
TK127 ORGANIC MAKE-UP TANK
AG103
AG104
TK102
TK101
TK105
EXTRACT MIXER SETTLER NO 1
PU111 A&B
TK112 RAFFINATE TANK
PU113 A & B
RAFFINATE TO MINE
STEAM
HX114 RAFFINATE HEATER
FOAM
FP131 SX FOAM FIRE SYSTEM
STEAM
WATER
BO133 BOILER
114
114 0
AG108
AG109
TK107
TK110
TK108 EXTRACT MIXER SETTLER NO 2
114
114 1.0
PREGNANT SOLUTION FROM MINE
FROM SETTLERS
PU125
TK126 CRUD TANK
114 1.0
AG116
TK115
TK117
STRIP MIXER SETTLER
PU120
SX AREA SUMP
8.33 30
SOLVENT EXTRACTION
BLEED
PU122 A & B
CO119 COALESCER
PU118 A & B
8.33 45
PU121 A&B
TK120 PREGNANT ELECTROLYTE TANK
CH234 ELECTROWINNING CIRCUIT CRANE
FN232 ACID MIST FAN
ME231 ACID MIST ELIMINATOR
SB235/6 STRONGBACKS
SF237 SHORTING FRAME
E/W CELL COVERS
WATER
TK201 COMMERCIAL RECIRCULATION TANK
PU235A/B
PU234A/B
35.26 33.5
BA233 PLASTIC BALLS
EC205 to 218 ELECTROWINNING CELLS
AN219 ANODES FOR EC201-214 PLUS S
CA220 CATHODES FOR EC201-214 PLUS SPARES
ER222 ELECTROWINNING CIRCUIT RECTIFIER
PU229 PORTABLE
H_2SO_4 BY TANKER
TK204 CONCRETE SULPHURIC ACID TANK
35.26 30
LOADED CATHODES
ELECTROWINNING
STRIPPED CATHODE BLANKS
LOADED CATHODES
PU230
ELECTROWINNING SUMP
KEY
m³/h g/L Cu
PU227
TK203 DUMP TANK
TK202 CATHODE WASH TANK
STEAM
CR221 CATHODE STRIPPING RACK
WS223 WEIGH SCALE FOR CATHODES
COPPER PRODUCT 3T/DAY
noranda
ADM
Acres Davy McKee
C 1030
S/X AND E/W PROCESS FLOW SHEET
A 03 01

Figure 4 - SX-EW Flowsheet and Equipment Diagram

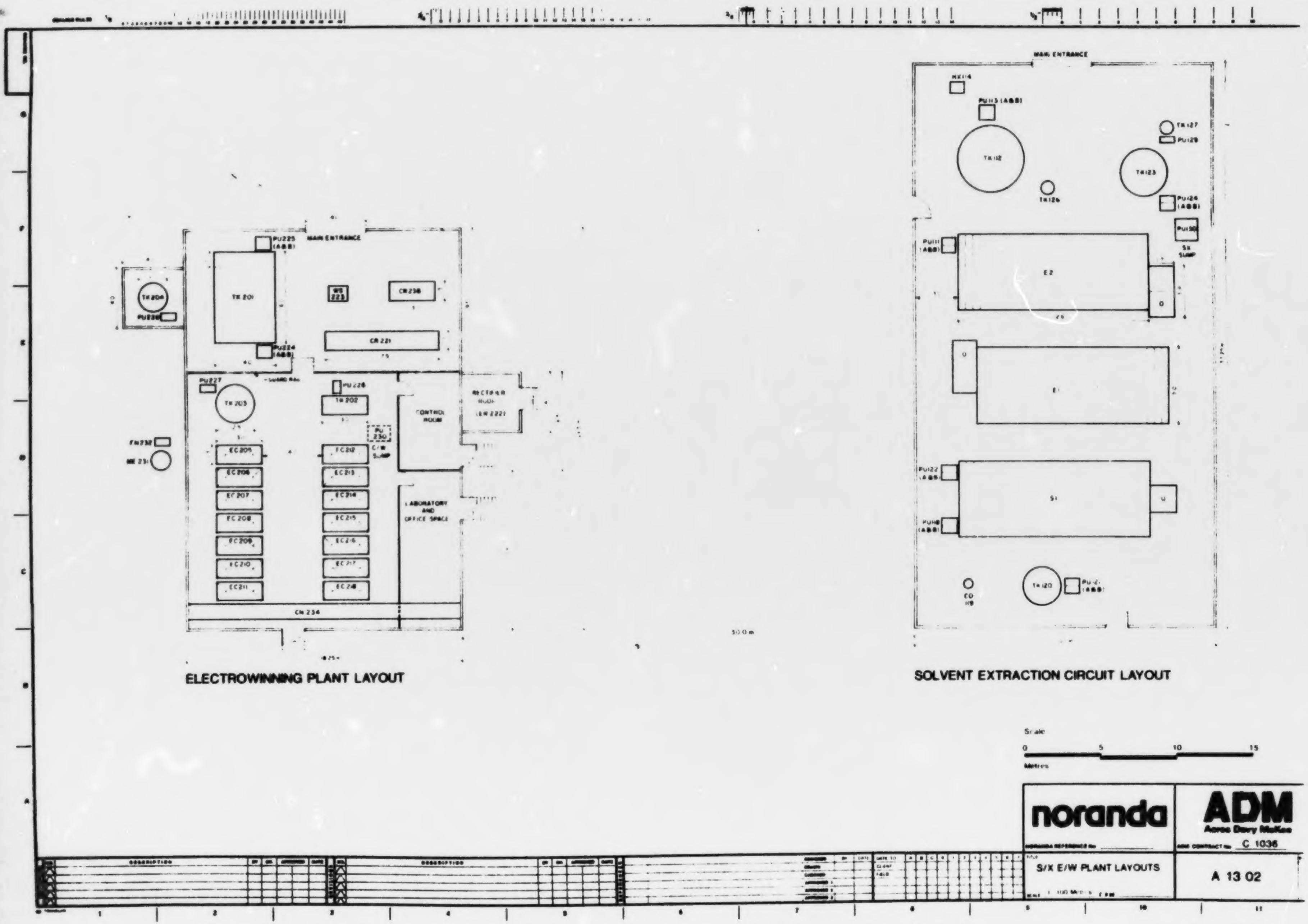
ELECTROWINNING PLANT LAYOUT
MAIN ENTRANCE
TK 201
TK 204
PU 225 (A&B)
PU 224 (A&B)
WS 223
CR 238
CR 221
PU 227
TK 203
PU 226
TK 202
PU 230
C/W SUMP
CONTROL ROOM
RECTIFIER ROOM (EW 222)
LABORATORY AND OFFICE SPACE
FN 232
ME 231
EC 205
EC 206
EC 207
EC 208
EC 209
EC 210
EC 211
EC 212
EC 213
EC 214
EC 215
EC 216
EC 217
EC 218
CN 234
SOLVENT EXTRACTION CIRCUIT LAYOUT
MAIN ENTRANCE
HX 114
PU 113 (A&B)
TK 112
TK 126
TK 127
PU 129
TK 123
PU 124 (A&B)
PU 130
SX SUMP
PU 111 (A&B)
E2
E1
S1
PU 122 (A&B)
PU 118 (A&B)
TK 120
PU 121 (A&B)
Scale
0
5
10
15
Metres
noranda
ADM
Acres Davy McKee
C 1036
S/X E/W PLANT LAYOUTS
A 13 02
DESCRIPTION

Figure 5 - SX-EW Plant Layout

to be progressively reduced (same operations in shorter time periods to recover decreasing copper tonnages) in order to maintain the operating costs closer to the period of full production. Since cementation has (i) less unit operations than SX-EW, (ii) requires less supervision and (iii) has a higher proportion of batch vs continuous activities, it will be easier to transform to a single shift operation and will adapt better to cost-cut reductions.

6.5 Labour Requirements

6.5.1 Cementation

Most of the activities of a cementation plant of this capacity could be handled by a single operator in one shift. The batch filtration operation can be carried out in two shifts (one operator in each) at the size and with the equipment costed in the capital cost estimate; but dead times will exist, and both the operators in the launders and the filter-presses can be used in other activities. The small plant scale will demand that operators perform different activities. In the case of the cementation alternative, in order to have one person in the recovery plant at all times, the operators will also perform duties in the laboratory and watches in the control room. The filter presses will be manned by two operators, with some functions in the maintenance department, to be divided with the operator of the leaching section. One supervisor has been included in the estimate of the recovery plant, but will also be responsible for the overall project.

Labour requirements for the cementation plant are as follows:

	No.Operators/ Shift	Number Shifts	Days/Week	Total Number Operators
Cementation (also laboratory, control room and leaching)	1	1	7	4
Filtration (maintenance)	1	2	5	2
Supervision (security, planning and leaching section)	1	1	5	1
	Total Recovery Plant			7
Leaching (and maintenance)	1	1	5	1
	Total in Project			8

6.5.2 Solvent Extraction-Electrowinning

As in the case of cementation, operators will be required to perform several activities in different unit operations. The SX circuit requires a closer supervision than the cementation launders, but still operators would have some dead time. It is proposed that they also perform routine analysis in the laboratory and watches in the control room, but unlike cementation, they will not be available for work in the leaching section.

In electrowinning, it will be necessary to have 1 operator per shift in the cellhouse, and part of his activities can be interchanged with personnel from the SX plant, to free an operator for additional work in the leach section.

Cathode stripping will be handled by two operators during one shift. One supervisor was included in the estimate of the recovery plant, but will have responsibilities for the whole project.

Labour requirements for the SX-EW plant are as follows:

	No.Operators/ Shift	Shifts/ Day	Days/Week	Total Number Operators
Solvent Extraction (laboratory, control room)	1	3	7	4
Cellhouse (and SX)	1	3	7	4
Cathode Stripping	2	1	7	3
Supervision	1	1	5	1
		Total in SX-EW Plant		12
Leaching	1	1	5	1
		Total Operators in Project		13

7. ECONOMICS

Noranda sub-contracted Acres Davy McKee (ADM) to cost the underground pumping and piping system and the cementation and SX-EW recovery plants. All costs are based upon 1985 Canadian dollars. Operating cost estimates for the leach plant and effluent treatment plant were not included in ADM's estimates and were carried out by Noranda Research Centre. A copy of ADM's report is included in the Appendix.

The recovery of copper by solvent extraction-crystallization was also costed by ADM and included in the report. This alternative was not considered in this evaluation since it does not appear to offer major advantages in recovery costs, and does not offset the high mining development costs estimated in Phase II of this study.

7.1 Capital Cost Estimate

7.1.1 Leaching Section

The solution distribution system drawing of the Geco orebody, and equipment requirements described in the previous section, were used to estimate the capital cost of the leaching section. The cost breakdown for the initial capital investment is given in Table XVIII. Additional annual capital costs for commissioning new stopes (up until year 10 of the project) are shown in Table XIX. Other plant components, costss such as the electrical installation, instrumentation, laboratory and control romm have been estimated using factors on the major equipment investment. A start-up cost equivalent to 10% of the fixed capital investment has been incorporated.

7.1.1.1 Estimate

The total capital cost for the leaching section is estimated to be \$1.64 M. This value contains a contingency allowance of 15%.

TABLE XVIII

CAPITAL COSTS OF LEACHING SECTION

Direct Costs*

			Cost $
1.	Piping, includes	a. Recovery Plant to Stopes b. Pregnant Solution to Surface c. Solution Distribution d. Sprinklers	475,000
2.	Pumps, includes	a. Solution Transfer Pumps to Surface 3 stages, 2-150 hp pumps each, one stand-by b. Solution Recirculation	233,000
3.	Bleed & Neutralization Circuit, includes	a. Neutralization Tank b. Solution transfer Pump c. Agitator, piping	40,000
4.	Valves		15,000
		Total Installed Equipment	763,000
5.	Pond for Neutralized Iron Residue ($5/tonne residue)		150,000
6.	Electrical (6% of Initial Cost)		46,000
7.	Instrumentation (4% of Initial Capital)		30,000
8.	Control Room (5% of Initial Capital)		38,000
9.	Laboratory (8% of Initial Capital)		61,000
			1,088,000
10.	Start-up Costs (10% of Fixed Capital Investment)		108,000
		Total Direct Costs	1,196,000

Indirect Costs

Field Indirect Costs (3% of Direct Costs)	36,000
Engineering and Construction Management (15% Direct Costs)	179,000
Construction Insurance 1 year	14,000
Contingency 15%	211,000
Total Indirect Costs	440,000
Total Capital Cost Leaching Section	$1,636,000

---

* Costs of sumps and drain ditches have been estimated in the Mine Development Section.

7.1.1.2 List of Exclusions

The following items were not included in capital cost estimate:

power lines or water supply to site. These are understood to be available from the mill and mine operations.

all sumps and drains. These have been considered as part of the mining development cost.

taxes or duties.

financing during construction.

working capital, spare parts or tools.

TABLE XIX

CAPITAL COSTS PER ADDITIONAL YEAR OF STOPE COMMISSION

Pipes and Couplings	$ 4,000
Pumps (2)	9,600
Sprinklers and Attachments	400
Valves	500
	$14,500
Contingency (15% of Direct Costs)	2,175
	$17,000

7.1.2 Recovery Plant

The plant drawings for cementation and solvent extraction shown in Figures 3 and 4, and equipment accounts described in the previous sections were used to estimate the capital cost of the recovery plant.

ADM's capital cost estimates were modified to exclude solution heating equipment, because it was estimated that the heat of reactions in leaching and the heat losses in the pumping of the PLS to surface would result in an average temperature of the pregnant solution of 22°C. Although the temperature is lower than desired for the operation of the SX circuit, it was found acceptable and resulted in significant cost reductions, particularly the operating costs.

7.1.2.1 Estimate

The total expected capital cost of a 1000 tonne/year copper recovery plant is \$1.73 M and \$5.09 M for the cementation and SX-EW alternatives (Tables XX and XXI), respectively. The cost breakdown for SX-EW is \$2.82 M for the solvent extraction section and \$2.27 M for the electrowinning section.

7.1.2.2 List of Exclusions

The following items were not included in capital cost estimate:

extensive site preparation

power lines and water supply to site

storage of cement copper or cathodes

taxes or duties

working capital or tools

start-up costs

financing during construction

7.2 Operating Costs

7.2.1 Leaching Section

Operating costs of the leaching section including bleed neutralization and residue disposal have been estimated at 12.03¢/lb of produced copper in the case of a SX-EW recovery plant and 12.67¢/lb copper for the alternative with a cementation plant. A summary of the operating costs for both alternatives is given in Table XXII.

TABLE XX

CAPITAL COST[1] OF COPPER CEMENTATION PLANT

(1000 tonnes/year)

Direct Costs	$
Sitework/earthwork	Included with building
Foundations and concrete	Included with building
Foundations and concrete	Included with building
Steel/structures	90,000
Buildings/enclosures	580,000
Equipment	370,000
Electrical	100,000
Instrumentation	25,000
Piping	75,000
Miscellaneous	25,000
Total Direct Costs	$1,265,000
Indirect Costs	
Field indirect costs	38,000
Engineering and construction management	190,000
Construction insurance	15,000
Contingency	224,000
Total Indirect Costs	467,000
Total Plant Cost	$1,732,000

[1] 1985 Cdn $

TABLE XXI

CAPITAL COST[1] OF COPPER SOLVENT EXTRACTION-ELECTROWINNING PLANT

(1000 tonnes/year Cu)

Direct Costs	SX Section	Electrowinning
	$	$
Sitework/earthwork	167,000	142,000
Foundations and concrete	377,000	194,000
Steel/structures	15,000	93,000
Buildings/enclosures	246,000	234,000
Equipment	777,000	500,000
Electrical	128,000	215,000
Instrumentation	50,000	100,000
Piping	250,000	110,000
Miscellaneous	50,000	50,000
Total Direct Costs	$2,060,000	$1,638,000
Indirect Costs		
Field indirect costs	61,800	60,000
Engineering and construction management	309,000	250,000
Construction insurance	24,000	23,000
Contingency	365,000	300,000
	759,000	633,000
Total Section Cost	$2,819,000	$2,271,000
Total Plant Cost $5.1 M		

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[1] 1985 Cdn $

TABLE XXII

OPERATING COST[1.] LEACHING SECTION

	SX-EW		CEMENTATION	
	$/year	¢/lb	$/year	¢/lb
Labour	45,000	2.05	45,000	2.05
Reagents	34,000	1.55	48,250	2.19
Supplies	5,500	0.25	5,500	0.25
Power	167,000	7.59	167,000	7.59
	251,000	11.44	266,000	12.08
Residue Disposal	13,000	0.59	13,000	0.59
Total Operating Cost	264,000	12.03	279,000	12.67

[1.] 1985 Cdn $

Major operating costs in the leaching section are the pumping of the pregnant leach solution to surface and reagent consumption. A labour component equivalent to one operator has been included in this estimate, although this operator would not be required full-time in this section (see section 7.2.1.1). The reagents cost vary according to which type of recovery plant is adopted. In cementation, a small amount of acid will be required to adjust the acidity of barren solution from approximately 0 to 0.3 gpL before it is transferred underground. In the SX-EW option, the acidity of the raffinate is sufficient for direct injection to the leaching section. A breakdown of the operating cost components is shown for both alternatives in Table XXIII.

TABLE XXIII

BREAKDOWN OF OPERATING COSTS[1] - LEACHING SECTION

Cost Item SX-EW		Cementation	
Reagents	$/year		$/year
H_2SO_4		215 t/y x $50/t	10,750
Lime for Bleed Neutralization		398 t/y x $\frac{1}{0.85}$ x \$80/t	37,500
362 t/y x $\frac{1}{0.85}$ x \$80/t =	34,000		48,250
Labour			
No. of operators/year = 1	45,000		45,000
Power			
Consumption 455 hp x 0.745 x $\frac{1}{0.65}$ x 8000 h/y = 4.17 x 10^6 kWh/y x 4¢/kWh	167,000		167,000
Supplies	5,500		5,500
Residue Disposal			
~1500 t/y(solids) x 2.86(t/m^3) x $3/t	13,000		13,000
Total Operating Cost $/year	269,500	$/year	279,000
¢/lb Cu	12.02	¢/lb Cu	12.67

[1] 1985 Cdn $

7.2.2 Recovery Plant

7.2.2.1 Cementation

The annual operating costs for the cementation plant were estimated to be \$0.86 M or 38.83¢/lb Cu (Table XXVII).

These costs have been modified from ADM's estimate to reflect an increase of scrap iron costs delivered to Geco from \$90/tonne to \$180/tonne and to exclude the energy requirements for solution heating.

TABLE XXIV

RECOVERY PLANT OPERATING COSTS[1]
CEMENTATION

	\$/year	¢/lb
Labour	270,000	12.25
Reagents	450,000	20.45
Supplies	62,000	2.81
Power	15,000	0.68
	797,000	36.19
Supervision and Planning	58,000	2.64
Total Operating Costs	856,000	38.83

[1] 1985 Cdn \$

7.2.2.2 Solvent Extraction-Electrowinning

The annual operating costs for the solvent extraction-electrowinning plant were estimated to be $0.86 M or 38.80¢/lb Cu. Details are presented in Table XXV.

TABLE XXV

RECOVERY PLANT OPERATING COSTS[1]

SX-EW

	¢/lb	$/year
Solvent Extraction		
Labour	8.16	180,000
Reagents	5.22	114,960
Supplies	1.00	22,050
Power	0.74	16,390
	15.12	334,400
Electrowinning		
Labour	14.29	315,000
Reagents	0.77	17,000
Supplies	1.81	40,000
Power	4.17	92,000
	21.04	464,000
Supervision & Planning	2.64	58,000
Total Operating Costs	38.80	856,000

[1] 1985 Cdn $

7.3 Cementation vs Solvent Extraction-Electrowinning

The total capital cost of this project (leaching and recovery) has been estimated to be \$3.4 M if copper is recovered by cementation, or \$6.7 M if solvent extraction-electrowinning is selected. Capital costs breakdown for the two options is shown in Table XXVI.

TABLE XXVI

CAPITAL COST COMPARISON OF CEMENTATION AND SX-EW

Leaching	\$1,636,000	Leaching	\$1,636,000
Cementation	1,731,000	SX	2,819,000
		Electrowinning	2,271,000
Capital Cost of Project	\$3,367,000		\$6,726,000

Total operating costs have been estimated at 50.83 ¢/lb and 51.52 ¢/lb for cementation and SX-EW, respectively. A breakdown of operating costs is given in Table XXVII. In the case of producing copper cement it is necessary to add costs of transportation of the cement to the smelter at \$38.50/tonne and smelting and refining charges of 13.18¢/lb (which includes transportation). In the SX-EW case, since high purity cathode is produced in a remote location such as Geco, it is necessary to add cathode transportation costs, estimated at 4.09¢/lb. In conclusion, the total cost of producing copper by cementation is 66.70¢/lb and in SX-EW, it is 54.92¢/lb.

Since the additional \$3.4 M capital cost required for the SX-EW option cannot be offset by the 11.81¢/lb lower production costs, and as discussed in section 5.6, the cementation plant offers significant technical advantages over SX-EW in adapting to variations in pregnant solution volume and lower copper production, the cementation option was selected for the overall project evaluation in Phase IV (mining and leaching).

TABLE XXVII

OPERATING COST COMPARISON OF CEMENTATION AND SX-EW

(¢/lb Cu)

COST ITEM	CEMENTATION		SX-EW
Leaching	12.10		11.44
Supervision and Planning	2.64		2.64
Recovery Plant - Cementation	36.19	SX	15.12
		Electrowinning	21.04
Residue Disposal	0.59		0.59
	51.52		50.83
Transportation of cement copper	2.06	Transportation of cathodes	4.09
Smelting and Refining charges	13.18		
Total Production Cost	66.8		54.9

8. FACTORS AFFECTING BACTERIAL LEACHING TECHNOLOGY

Since no experimental data exists for this deposit, many important design parameter values were assumed based upon our criteria and available literature. Even after the optimistic assumptions that (i) a 55% copper recovery can be obtained within 10 years from a 6-in. diameter chalcopyrite ore and (ii) sufficient permeability exists within the blasted stopes to allow the 10 $L/m^2/h$ solution application rate, the following additional parameters are considered critical to the success of the project.

8.1 Oxygen Availability

It was assumed that there was enough O_2 in the stopes, between oxygen-carrying leach solution and diffusivity of air into the void spaces of the broken ore in the stopes, to allow proper oxidation of the sulphides in the presence of bacteria. This assumption was taken because all mine-out areas of the deposit are connected wto the mine ventilation system. If this were not the case, air injection into the lower part of the stope would be required at an additional cost.

8.2 Mineralogy

The presence of higher sulphide contents could lead to unacceptable high temperatures (>45°C) within the stopes being leached at the rates of solution application considered for this study. This would result in a decrease of bacterial activity and an interruption of the leaching process until the process re-equilibrated.

It was also assumed that the ore contains a sufficient quantity of acid consuming material to neutralize the acid produced by oxidation of sulphides. In addition to unacceptably high temperatures, therefore, the presence of higher sulphide content and/or lower acid consuming gangue would also result in low pH values which would cease of decrease bacterial activity.

8.3 Iron Precipitation

It was assumed that all of the iron introduced into solution both by the dissolution of sulphide minerals and cementation of copper, precipitate as basic iron sulphate within the stopes to maintain an equilibrium iron concentration of approximately 4 gpL. It was assumed that the iron precipitate would not coat the ore and therefore interfere with leaching rate or create channeling. If this were not the case, a volume of solution equivalent to the iron introduced would require to be bled and treated in an effluent treatment pond at an additional cost of 16¢/lb Cu.

8.4 Leaching Temperature

Rock temperature at Geco, as in most Canadian mines, is 15°C. Solutions to and from the recovery plant are to be transferred through the mine shaft. The heat balance calculated shows that no external solution heating is required in the form of a submerged combustor. This fact alone decreases the operating costs by ∿20¢/lb copper.

8.5 Production Slowdown

This is one of the most overlooked areas of chemical mining. Once the last stopes have been commissioned for leaching, the project enters a period of reduced recoveries. This can be seen in Tables VI to VIII. In order to continue copper production, at average costs, it is necessary to accommodate all the unit operations to the progressively smaller tonnages. This means reducing labour, rescheduling shifts and having personnel working in several different tasks. The decision on the type of recovery process, as discussed in section 6.4 must also be made considering the last years of operation.

It is very likely that, based upon economic considerations, the project would be shut down before the end of the projected schedule. This means that there will be less than the 55% copper extraction from the last groups of stopes to be commissioned.

9. RECOMMENDATIONS FOR COST REDUCTION

The optimization of the leaching and recovery process is outside the scope of this project. The cementation process was selected on the basis of economical and practical grounds. One change that could significantly reduce costs would be to install the cementation launders underground and haul the cement copper to surface. It is worth noting that 87% of the major equipment costs and 66% of the operating costs are dedicated to pumping the 1 gpL copper solution to surface.

The cementation plant for 1000 tonnes/year copper production was estimated to be 23 m wide x 30 m long x 6 m high and could be installed within a worked out area of two stopes.

APPENDIX

ADM COST ESTIMATES

4.2 - Raffinate

Flow rate	114 m^3/h
Temperature	35°C
Method of discharge	In pipeline

5 - ENVIRONMENTAL CONDITIONS

5.1 - Location

The plant will be located at the Geco Mine in northwestern Ontario.

5.2 - Design Temperature

Mean daily maximum temperature = 6.9°C
Extreme daily maximum temperature = 39.4°C
Mean daily minimum temperature = -4.7°C
Extreme daily minimum temperature = -42.2°C

5.3 - Precipitation

Average annual total precipitation = 861.7 mm
Average annual snowfall = 309.5 mm

5.4 - Site Conditions

The site is assumed to be relatively level, clear and that a minimum of excavation with no rock excavation requirement and backfilling is required.

5.5 - Units

The plant has been designed using metric units.

DESIGN CRITERIA

1 - SOLVENT EXTRACTION/ELECTROWINNING

1.1 - Solvent Extraction

(a) Extraction

Pregnant solution - flow rate = 114 m^3/h
- Cu conc. = 1 g/L
- Fe conc. = 3 to 4 g/L
- pH = 2 to 2.5
- temperature = 15°C

No. of stages = 2
Mixer retention time/stage = 2 min
No. of mixer boxes/stage = 2
Specific settling rate = 4 m^3/m^2h (total flow)
Overall O:A ratio = 1:1
Operating O:A ratio = 1:1
Reagent concentration in organic = 3 to 5%

(b) Stripping

No. of stages = 1
Mixer retention time/stage = 1 min
No. of mixer boxes/stage = 1
Specific settling rate = 4 m^3/m^2h (total flow)
Overall O:A ratio = 13.7:1
Operating O:A ratio = 1:1
Pregnant electrolyte - flow rate = 8.33 L/min
- Cu conc. = 45 g/L

Spent electrolyte - Cu conc. = 30 g/L
- H_2SO_4 conc. = 170 to 180 g/L

1.2 - Electrowinning

Stripping method	Total production stripping
Current density	215 A/m^2
Current efficiency	85%
Cathode size	1 m x 1 m
Cathode spacing	100 mm
Cathodes/cell	21
Cathodes	S.S. blanks
Anodes	Lead/calcium/tin

2 - CEMENTATION

Cementation equipment	Launders
Iron consumption	2.5 kg/kg Cu
Copper recovery	97%
Cement composition	85% Cu
Cement filter type	Plate and frame

PROCESS DESCRIPTION

The different processes considered in this study for recovering copper from the pregnant solution are as follows

- solvent extraction/electrowinning
- cementation
- solvent extraction/crystallization.

(a) Solvent Extraction/Electrowinning

With this method, the pregnant solution from underground is fed to the solvent extraction circuit. Here, the copper is extracted from the pregnant solution by countercurrent contact with organic in the two extraction mixer settlers. The raffinate from the extract two mixer settler is heated to 35°C and returned underground for leaching. The loaded organic from the extract one mixer settler is sent to the strip mixer settler where the copper is removed by contact with spent electrolyte containing 170 to 180 g/L H_2SO_4. The stripped organic is then recirculated to the extraction mixer settlers and the pregnant electrolyte is sent to the electrowinning circuit.

The electrowinning circuit has been designed for total production stripping. The pregnant electrolyte is fed to the commercial recirculation tank from where it is recirculated to the 14 commercial electrowinning cells. The spent electrolyte from the cells is returned to the solvent extraction circuit. A 7-day cycle is used for deposition on the cathodes. Everyday, 42 cathodes are removed from the cells and the copper deposits are stripped off, weighed and packaged ready for transportation.

The quality of the copper product is high and it can be sold on the LME or any other exchange.

(b) Cementation

With this method, the pregnant solution from underground is fed to a series of launders containing de-tinned cans. The copper is cemented out of the pregnant solution by the iron and the barren solution is heated to 35°C and recirculated underground.

The launders have a false bottom on which the cans sit. Periodically one of the launders is taken out of service and the cement copper is washed off the cans. The cement passes through the false bottom and it is collected in a settling tank. The solution is decanted off and then the cement is removed from the settling tank by a crane with a clam shell. The crane can also be fitted with an electromagnet and it is used to load cans into the launders. The settled cement is then reslurried and pumped to a plate and frame filter. The cement filter cake containing 15 to 20% moisture is transported to a smelter. The product contains only 85% copper and so it has to be sent to a smelter to produce a high grade product.

(c) Solvent Extraction/Crystallization

This is an alternative to the solvent extraction/electrowinning method described above. With this method the pregnant strip solution from solvent extraction is fed to a crystallization circuit where copper sulfate crystals are produced.

The pregnant strip solution is first fed to an evaporator and then to a crystallizer. The crystals discharged from the crystallizer are sent to a settling tank whose underflow is fed to a centrifuge. The moisture content of the crystals is 2 to 3% and so they are fed to a dryer to reduce the moisture content to 0.5%. Then the crystals are drummed ready for shipping.

The product is copper sulfate crystals which are of a good grade and can be sold directly.

EQUIPMENT LISTS

SOLVENT EXTRACTION

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
TK101	Extract 1 Mixer Box No. 1	Size = 1.7 m x 1.7 m x 1.7 m high	Supplied with draft tube, top baffle and cover. FRP	
TK102	Extract 1 Mixer Box No. 2	Size = 1.7 m x 1.7 m x 1.7 m high	Supplied with top baffle and cover. FRP	
AG103	Extract 1 Pump Mixer	Diameter = 0.7 m	Double-shrouded with 8 back-swept vanes. 316L S.S.	3.7
AG104	Extract 1 Agitator	Diameter = 0.6 m	Radial flow turbine. 316L S.S.	2.2
TK105	Extract 1 Settler	Size = 5.0 m wide x 12.6 m long x 1.2 m high	Supplied with introductory slot, picket fence, organic weir, aqueous weir and cover. FRP	
TK106	Extract 2 Mixer Box No. 1	Size = 1.7 m x 1.7 m x 1.7 m high	Supplied wtih draft tube top baffle and cover. FRP	
TK107	Extract 2 Mixer Box No. 2	Size - 1.7 m x 1.7 m x 1.7 m high	Supplied with top baffle and cover. FRP	
AG108	Extract 2 Pump Mixer	Diameter = 0.7 m	Double-shrouded with 8 back-swept vanes. 316L S.S.	3.7
AG109	Extract 2 Agitator	Diameter = 0.6 m	Radial flow turbine. 316L S.S.	2.2

SOLVENT EXTRACTION - 2

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
TK110	Extract 2 Settler	Size = 5.0 m wide x 12.6 m long x 1.2 m high	Supplied with introductory slot, picket fence, organic weir, aqueous weir and cover. FRP	
PU111 A & B	Raffinate Transfer Pumps	Flow = 114 m^3/h Head = 10 m Size = 6 x 4 x 13	Horizontal centrifugal 316L S.S.	5.5
TK112	Raffinate Tank	Size = 5.5 m dia. x 5.5 m high	Supplied with cover. FRP	
PU113 A & B	Raffinate Feed Pump	Flow = 114 m^3/h Head = 15 m Size = 6 x 4 x 13	Horizontal centrifugal 316L S.S.	7.5
HX114	Raffinate Heater	Heat required = 2.3 Million KCal/H	Steam heated plate exchanger 316L S.S.	
TK115	Strip Mixer Box	Size = 1.7 m x 1.7 m x 1.7 m high	Supplied with draft tube, top baffle and cover. FRP	
AG116	Strip Pump Mixer	Diameter = 0.7 m	Double-shrouded with 8 back-swept vanes. 316L S.S.	3.7
TK117	Strip Settler	Size = 5 m wide x 12.6 m long x 1.2 m high	Supplied with introductory slot, picket fence, organic weir, aqueous weir and cover. FRP	

SOLVENT EXTRACTION - 3

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
PU118 A & B	Pregnant Electrolyte Transfer Pump	Flow = 8.33 m^3/h Head = 35 m Size = 1 1/2 x 1 x 6	Horizontal centrifugal 316L S.S.	2.2
CO119	Pregnant Electrolyte Coalescer	Capacity = 8.3 m^3/h	Dual media sand filter	
TK120	Pregnant Electrolyte Tank	Size = 2.5 m dia. x 2.5 m high	Supplied with cover. FRP	
PU121 A & B	Pregnant Electrolyte Feed Pump	Flow = 8.33 m^3/h Head = 10 m Size = 1 1/2 x 1 x 6	Horizontal centrifugal 316L S.S.	0.55
PU122 A & B	Stripped Organic Transfer Pump	Flow = 114 m^3/h Head = 10 m Size = 6 x 4 x 13	Horizontal centrifugal 316L S.S.	5.5
TK123	Stripped Organic Tank	Size = 3 m dia. x 3 m high	Supplied with cover. FRP	
PU124 A & B	Stripped Organic Feed Pump	Flow = 114 m^3/h Head = 10 m Size = 6 x 4 x 13	Horizontal centrifugal 316L S.S.	5.5
PU125	Portable Crud Removal Pump	Flow = 8.3 m^3/h Head = 10 m Size = 1 1/2 x 1 x 6	Horizontal centrifugal 316L S.S.	0.55

SOLVENT EXTRACTION - 4

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
TK126	Crud Tank	Size = 1 m dia. x 2 m straight side with 45° conical bottom	FRP	
TK127	Organic Make-Up Tank	Size = 1 m dia. x 1 m high	Supplied with baffles. FRP	
AG128	Organic Make-Up Tank Agitator	Diameter = 0.3 m	316L S.S.	0.18
PU129	Organic Make-Up Pump	Size = 1 1/2 x 1 x 6	Horizontal centrifugal 316L S.S.	0.55
PU130	SX Area Sump Pump	Size = 1 1/2"	Vertical submerged centrifugal. 316L S.S.	0.55
FP131	SX Fire System		System complete with Co$_2$ and foam storage, distribution system, etc.	
01132	Organic Inventory	Reagent = 5 m^3 Diluent = 45 m^3	Reagent supplied as 60% solution in diluent. Diluent is high flash point kerosene.	
B0133	Boiler	Capacity - 5000 kg/h	Package boiler	3.7

ELECTROWINNING CIRCUIT

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
TK201	Commercial Recirculation Tank	Size = 6 m long x 4 m wide x 3 m high Internal baffled compartment of length 1.25 m, width 4 m, ht 2 m	FRP Supplied with cover	
TK202	Cathode Wash Tank	Size = 3 m long x 1.3 m wide x 2 m high	FRP Supplied with cathode support frame and sprays on the sides and cover	
TK203	Dump Tank	Size = 2.5 m dia x 2 m high	FRP with cover	
TK204	Conc. Sulphuric Acid Tank	Size = 2 m dia x 2 m high	FRP with cover	
EC205 to 218	Electrowinning Cells	Size = 3 m long x 1.3 m wide x 1.4 m high	Concrete walls with PVC paraliners covered with PVC sheets	
AN219	Anodes for EC201-214 plus Spares	Size = 1 m x 1 m Number = 330	Lead - calcium - tin	
CA220	Cathodes for EC201-214 plus Spares	Size 1 m x 1 m Number = 330	Stainless steel blanks	
CR221	Cathode Stripping Rack	Size = 7.5 m long x 1.3 m wide x 2.1 m high	Rubber lined steel frame	

ELECTROWINNING CIRCUIT - 2

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
ER222	Electrowinning Circuit Rectifier	9100 Amps 20-35 Volt Settings 6 Pulse		
WS223	Weigh Scale for cathodes	Scale Reading 0-220 kg	Suitable for operating in acidic environment	
PU224A/B	Electrolyte Circulation Pumps	Flow = 36 m^3/h Head = 10 m Size = 3 x 1 1/2 x 6	Horizontal Centrifugal 316L S.S.	
PU225A/B	Barren Electrolyte Pumps	Flow = 8.3 m^3/h Head = 10 m Size = 1 1/2 x 1 x 6	Horizontal Centrifugal 316L S.S.	0.55
PU226	Cathode Wash Pump	Flow 9.75 m^3/h Head = 35 m Size = 3 x 1 1/2 x 6	Horizontal Centrifugal 316L S.S	3.75
PU227	Dump Solution Pump	Flow = 11.7 m^3/h Head = 6 m Size = 1 1/2 x 1 x 6	Horizontal Centrifugal 316L S.S.	0.55
PU228	Sulphuric Acid Pump	Flow = 0.4 m^3/h Head = 5 m Size = 2 1/4" with 1/4" plunger	Duplex type pump Stainless steel - alloy	0.18
PU229	Portable Sludge Pump	Flow = 1 m^3/h Head = 5 m Size = 1 1/2 x 1 x 6	Horizontal Centrifugal, Portable (self-priming) 316L S.S.	0.25

ELECTROWINNING CIRCUIT - 3

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
PU230	Electrowinning Sump Pump	Size = 1 1/2	Vertical submerged centrifugal 316L S.S.	0.75
ME231	Acid Mist Eliminator	Capacity = 5500 L/min	FRP	
FN232	Acid Mist Fan	Capacity = 5500 L/min	Stainless Steel	0.25
BA233	Plastic Balls	Dia. = 20 mm No. Reqd. = 200,000	Polypropylene	
CN234	Electrowinning Circuit Crane	2 Tonne duty	Acid Resistant Pendant operated	
SB235/6	Strongbacks	Size = 3.5 m long x 1.8 m wide x 15 mm thick	Rubber lined steel frame	
SF237	Shorting Frame	Size = 3 m long x 1.3 m wide x 15 mm thick	Acid Resistant Steel	
CR238	Cathode Rack	Size = 3 m long x 1.3 m wide x 2.1 m high	Rubber lined steel frame	

CEMENTATION

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
CC301 to 306	Cementation Cell	Size = 2.4 m wide x 7.3 m long x 1.5 m deep	Acid resistant concrete cell with feed and overflow launders, central dividing wall, false bottom made of drilled plywood supported on 316 S.S. grids which rest on timbers, concrete floor below false bottom which slopes to 12 in. drain valve.	5.5
ST307 to 309	Settling Tank	Size = 5 m wide x 5 m long x 1.2 m deep	Acid resistant concrete sump with sloping bottom.	
PU310 A & B	Raffinate Transfer Pumps	Flow = 114 m^3/h Head = 10 m Size = 6 x 4 x 13	Horizontal centrifugal, 316L S.S.	5.5
TK311	Raffinate Tank	Size = 5.5 m dia. x 5.5 m high	Suppliers with cover. FRP.	
PU312 A & B	Raffinate Feed Pump	Flow = 114 m^3/h Head = 15 m Size = 6 x 4 x 13	Horizontal centrifugal, 316L S.S.	7.5
HX313	Raffinate Heater	Heat Reqd = 2.3 Million KCal/h	Steam heated plate exchanger, 316L S.S.	
PU314	Cementation Wash Pump	Flow = 45 m^3/h Head = 70 m Size = 3 x 12 x 8-1/2	Horizontal centrifugal, 316L S.S.	15.0

Cementation - 2

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
PU315	Cementation Decant Pump	Flow = 45 m^3/h Head = 10 m Size = 2-1/2 in.	Vertical submerged centrifugal, 316L S.S.	7.5
CR316	Crane	Capacity = 5T.	Supplied with 1.5-m dia. electromagnet and clamshell. Diesel-driven crawler.	
TK317	Filter Feed Tank	Size = 3.4 m dia. x 4.5 m high	Supplied with 4 baffles. FRP.	
AG318	Filter Feed Tank Agitator	Diameter = 1.13 m	RLCI.	
TK319	Copper Cement Hopper	Size = 1.6 m dia. x 2 m high	Stainless steel with conical bottom.	
PU320	Filter Feed Pump	Pressure = 1550 kPa	Abel piston membrane type.	5.5
FP321	Filter Press	No. of Chambers = 30 Size = 4.3 m wide x 1.1 m long x 1.83 m high	Supplied complete with drip trays and light curtains.	2.2
TK322	Filter Cake Tray	Size = 6 m wide x 2 m long x 1 m high	Stainless steel.	
TK323	Boiler	Capacity = 5000 kg/h	Package boiler.	3.7

CAPITAL COST ESTIMATES

1 - SOLVENT EXTRACTION/ELECTROWINNING

(a) SX

Direct Costs	Can $
Sitework/earthwork	167,000
Foundations and concrete	377,000
Steel/structures	15,000
Buildings/enclosures	246,000
Equipment	852,000
Electrical	128,000
Instrumentation	50,000
Piping	250,000
Miscellaneous	50,000
Total Direct Costs	2,135,000
Indirect Costs	
Field indirect costs	63,000
Engineering and construction management	320,000
Construction insurance	25,000
Contingency	375,000
Total Indirect Costs	783,000
Total Plant Cost	2,918,000

(b) EW

Direct Costs	**Can $**
Sitework/earthwork	142,000
Foundations and concrete	194,000
Steel/structures	93,000
Building/enclosures	234,000
Equipment	500,000
Electrical	215,000
Instrumentation	100,000
Piping	110,000
Miscellaneous	50,000
Total Direct Costs	1,638,000
Indirect Costs	
Field indirect costs	60,000
Engineering and construction management	250,000
Construction insurance	23,000
Contingency	300,000
Total Indirect Costs	633,000
Total Plant Cost	2,271,000

2 - CEMENTATION

Direct Costs	Can $
Sitework/earthworth	Included with building
Foundations and concrete	Included with building
Foundations and concrete	Included with building
Steel/structures	90,000
Buildings/enclosures	580,000
Equipment	445,000
Electrical	100,000
Instrumentation	25,000
Piping	75,000
Miscellaneous	25,000
Total Direct Costs	1,340,000
Indirect Costs	
Field indirect costs	40,000
Engineering and construction management	200,000
Construction insurance	16,000
Contingency	240,000
Total Indirect Costs	496,000
Total Plant Cost	1,836,000

Notes

1. The capital cost estimates are of an order of magnitude type with an overall accuracy of +25%. All costs are expressed in Canadian dollars as of March 1985. Costs and prices received in US dollars have been converted to Canadian dollars at the rate of Can $1.40 per US dollar.

2. The estimates have been produced by developing preliminary specifications for major equipment and preliminary take-offs for foundation, buildings, piping, etc. Pricing information was then obtained from either vendor budget type quotes or in-house information.

OPERATING COST ESTIMATES

1 - SOLVENT EXTRACTION/ELECTROWINNING

(a) SX

Summary

Area	¢/lb Cu	$/yr
Labor	8.16	180,000
Reagents	5.22	114,960
Supplies	1.00	22,050
Utilities	16.69	368,000
Power	0.74	16,390
Total	31.81	701,400

Detailed breakdown of each area of the operating cost is presented below.

Labor

Classification	No. Req'd	Monthly Rate	¢/lb Cu	$/yr
Operator	4	$3,750	8.16	180,000

Reagents

Reagent	Consumption lb/lb	Unit Cost ¢/lb	¢/lb Cu	$/yr
Organic	0.0042	8.4	3.53	77,792
Diluent	0.067	0.252	1.69	37,169
Total			5.22	114,961

Power

Consumption kWh/lb Cu	Unit Cost ¢/kWh	¢/lb Cu	$/yr
0.186	4	0.74	16,390

(b) EW

Summary

Area	¢/lb Cu	$/Yr
Labor	14.29	315,000
Reagents	0.77	17,000
Supplies	1.81	40,000
Power	4.17	92,000
Total	21.04	464,000

Detailed breakdown of each area of the operating cost is presented below.

Labor

Classification	No. Req'd	Monthly Rate	¢/lb Cu	$/yr
Operator	7	3,750	14.29	315,000

Reagents

Reagent	Consumption lb/lb	Unit Cost ¢/lb	¢/lb Cu	$/yr
Sulfuric acid	0.127	5.0 [1]	0.635	14,000
Cobalt sulfate	(est.)		0.136	3,000
Total			0.771	17,000

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[1] Noranda's cost estimate for sulphuric acid differs from ADM's estimate but does not significantly effect overall costs.

Power

Consumption kWh/lb Cu	Unit Cost ¢/kWh	¢/lb Cu	$/yr
1.04	4	4.17	92,000

2 - CEMENTATION

Summary

Area	¢/lb Cu	$/yr
Labor	12.25	270,000
Reagents	10.20	225,000
Supplies	2.81	62,000
Power	0.68	15,000
Utilities	16.69	368,000
Total	42.63	940,000

Detailed breakdown of each area of the operating cost is presented below.

Labor

Classification	No. Req'd	Monthly Rate	¢/lb Cu	$/yr
Operator	6	$3,750	12.25	270,000

Reagents

Regent	Consumption lb/lb	Unit Cost ¢/lb	¢/lb Cu	$/yr
Tin cans	2.5	4.08	10.2	225,000

Utilities

Type	Consumption m^3/lb Cu	Unit Cost ¢/m^3	¢/lbCu	$/yr
Natural gas	1.04	16	16.69	368,000

Power

Consumption kWh/lb Cu	Unit Cost ¢/kWh	¢/lb Cu	$/yr
0.17	4	0.68	15,000

Notes

1. Costs are in Canadian dollars and are based on a copper production of 1,000 t per year.
2. Monthly rates for labor include a mark up of 25% for fringe benefits.
3. The potential impact on bacterial leaching efficiency of high iron concentration in the raffinate from the cementation circuit has not been incorporated into the operating cost estimate. It is likely that large volumes of solution will have to be bled from the system to control iron concentration.

SIZE/COST RELATIONSHIPS

1 - SOLVENT EXTRACTION/ELECTROWINNING

(a) SX

Capital Cost

Over the range of annual copper production rates of 500 to 4000 t, the number of equipment items will be unchanged, but each item will vary in size in relation to the plant capacity.

The cost of different capacity SX circuits can be related by the following formula

$$\frac{\text{Capital Cost 1}}{\text{Capital Cost 2}} = \left(\frac{\text{Plant Capacity 1}}{\text{Plant Capacity 2}}\right)^{0.7}$$

Operating Cost

Over the range of annual copper production rates considered, the reagent, supplies, utilities and power costs will remain unchanged in cents per pound and the labor cost will vary in direct proportion to the plant capacity.

(b) EW

Capital Cost

The number of electrowinning cells and ancillaries required is directly proportional to the annual copper production rate. The cathode stripping circuit will be relatively unchanged over the range of annual copper production rates considered.

The cost of different EW circuits can be related by the following formula

$$\frac{\text{Capital Cost 1}}{\text{Capital Cost 2}} = \left(\frac{\text{Plant Capacity 1}}{\text{Plant Capacity 2}}\right)^{0.8}$$

Operating Cost

Over the range of annual copper production rates considered, the reagent, supplies and power costs will remain unchanged in cents per pound and the labor cost will vary in direct proportion to the plant capacity.

2 - CEMENTATION

Capital Cost

The number of launders and settling tanks required is directly proportional to the annual copper production rate. The cement filtration circuit will vary in size in relation to the plant capacity.

The cost of different cementation circuits can be related by the following formula

$$\frac{\text{Capital Cost 1}}{\text{Capital Cost 2}} = \left(\frac{\text{Plant Capacity 1}}{\text{Plant Capacity 2}}\right)^{0.8}$$

Operating Cost

Over the range of annual copper production rates considered, the reagents, supplies, utilities and power costs will remain unchanged in cents per pound and the labor cost will vary in direct proportion to the plant capacity.

ACRES DAVY McKEE ENGINEERING INC.

Toronto, Ontario

PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT

DSS File No. 07SQ.23440-4-9071

Submitted to: Centre de Recherche Noranda

Dated: March 8, 1985

ADM Eng. Ref.: C1036.00

ADM
Acres Davy McKee

Engineers and Managers

March 7, 1985
C1036.00

Centre de Recherche Noranda
240 boulevard Hymus
Pointe Claire, Quebec
H9R 1G5

Attention: Mr. A. Ismay

Dear Sir

Pre-Feasibility Analysis of
Bacterially-Assisted Underground
Leaching of a Canadian Copper
Sulphide Deposit
DSS File No. 07SQ.23440-4-9071

We enclose the balance of our information relating to the above project.

Crystallization

- Design Criteria
- Equipment List
- Capital Cost Estimate
- Operating Cost Estimate

Underground Pumping and Piping

- Description
- Capital Cost Estimate

Please let us know if you need our assistance in carrying out the sensitivity analysis and assembling the final report.

Yours very truly,

David J King

D.J. King, P.Eng.
Project Manager

DJK:smj
Enclosure

ACRES DAVY McKEE ENGINEERING INC.
480 University Avenue, Toronto, Ontario M5G 1V2
Telephone 416-595-2170 Telex 06-217815

1 - DESIGN CRITERIA
CRYSTALLIZATION

DESIGN CRITERIA

CRYSTALLIZATION

Crystallization feed rate = 8.33 m^3/h
Crystallization feed Cu conc. = 45 g/L
Crystallization discharge Cu Conc. = 30 g/L
Settling tank underflow = 50% solids
Centrifuge lake moisture content = 2 - 3%
Final product moisture content = 0.5%

2 - EQUIPMENT LIST
CRYSTALLIZATION

CRYSTALLIZATION

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
	Evaporator	Feed rate = 8.33 m^3/h	Rising film evaporator, Karbate heat exchanger and RLCS vapor head, etc.	
	Crystallizer	Production rate = 590 kg/h	Surface cooled crystallizer, telfon heat exchanger, FRP or RLCS tank.	18.7
	Boiler	Capacity = 3500 kg/h	Package boiler.	
	Cooling Tower	Flow = 245 m^3/h Inlet temp. = 29.4°C Outlet temp. = 25.6°C		
	Cooling Water Pump	Flow = 245 m^3/h Head = 20 m Size = 8 x 6 x 13	Horizontal centrifugal, C.I.	18.7
	Chilling System	Capacity = 238 000 Kcal/h	23% calcium chloride circulation system.	
	Crystallizer Discharge Pump	Flow = 6.4 m^3/h Head = 10 m Size = 1-1/2 x 1 x 6	Horizontal centrifugal, RLCI	0.55
	Settling Tank	Size = 1.8 m dia x 3 m total height	Cylindrical tank with conical bottom, FRP.	
	Settling Tank Underflow Pump	Capacity = 1 m^3/h Head = 10 m Size = 1-1/2 in.	ODS Pump, 316L S.S.	

Crystallization - 2

Equipment No.	Description	Size/Capacity	Type/Materials	Motor kW
	Centrifuge	Capacity = 590 kg/h crystals	Pusher centrifuge, 316 S.S.	11
	Spent Solution Tank	Size = 2.5 m dia x 2.5 m length	FRP.	
	Spent Solution Pump A & B	Flow = 8.33 m^3/h Head = 10 m Size = 1-1/2 x 1 x 6	Horizontal centrifugal 316L S.S.	0.55
	Conc. Sulphuric Acid Tank	Size = 3.2 m dia x 3.2 m high	FRP.	
	Conc. Sulphuric Acid Pump	Flow = 0.4 m^3/h Head = 5 m Size = 2-1/4 in. with 1/4 in. plunger	Duplex type pump. Stainless steel alloy.	0.18
	Copper Sulphate Dryer	Capacity = 590 kg/h		7.5
	Copper Sulphate Weighing and Drumming System	Capacity = 3 t in 8 h shift Drum capacity = 225 kg	Manually operated system including hopper, weight scale, etc.	
	Crystallization Area Sump Pump	Size = 1-1/2	Vertical submerged centrifugal, 316L S.S.	0.75

3 - CAPITAL COST ESTIMATE
CRYSTALLIZATION

CAPITAL COST ESTIMATES

CRYSTALLIZATION

Direct Costs	Can $
Sitework/earthworth	86,000
Foundations and concrete	128,000
Steel/structures	25,000
Buildings/enclosures	84,000
Equipment	1,039,000
Electrical	138,000
Instrumentation	30,000
Piping	125,000
Miscellaneous	25,000
Total Direct Costs	1,680,000
Indirect Costs	
Field indirect costs	50,000
Engineering and construction management	260,000
Construction insurance	20,000
Contingency	305,000
Total Indirect Costs	635,000
Total Plant Cost	2,315,000

ADM

Notes

1. The capital cost estimates are of an order of magnitude type with an overall accuracy of +25%. All costs are expressed in Canadian dollars as of March 1985. Costs and prices received in US dollars have been converted to Canadian dollars at the rate of Can $1.40 per US dollar.

2. The estimates have been produced by developing preliminary specifications for major equipment and preliminary take-offs for foundation, buildings, piping, etc. Pricing information was then obtained from either vendor budget type quotes or in-house information.

4 - OPERATING COST ESTIMATE
CRYSTALLIZATION

OPERATING COST ESTIMATE

CRYSTALLIZATION

Summary

Area	¢/lb Cu	$/yr
Labor	10.20	225,000
Reagents	7.70	167,790
Supplies	1.00	22,050
Power	0.88	19,400
Utilities	11.70	258,090
Total	31.48	692,330

Detailed breakdown of each area of the operating cost is presented below.

Labor

Classification	No. Req'd	Monthly Rate	¢/lb Cu	$/yr
Operator	5	$3,750	10.20	225,000

Reagents

Reagent	Consumption lb/lb	Unit Cost ¢/lb	¢/lb Cu	$/yr
H_2SO_4	1.54	5.00	7.70	167,790

Utilities

Type	Consumption m^3/lb Cu	Unit Cost ¢/m^3	¢/lbCu	$/yr
Natural gas	0.73	16	11.70	258,090

Power

Consumption kWh/lb Cu	Unit Cost ¢/kWh	¢/lb Cu	$/yr
0.22	4	0.88	19,400

Notes

1. Costs are in Canadian dollars and are based on a copper production of 1,000 t per year.

2. Monthly rates for labor include a mark up of 25% for fringe benefits.

3. The potential impact on bacterial leaching efficiency of high iron concentration in the raffinate from the cementation circuit has not been incorporated into the operating cost estimate. It is likely that large volumes of solution will have to be bled from the system to control iron concentration.

5 - DESCRIPTION

UNDERGROUND PUMPING & PIPING

UNDERGROUND PUMPING AND PIPING

DESCRIPTION

The barren solution from the SX plant is transported in a 150-mm rubber lined, carbon steel pipe down No. 4 shaft and along the development drifts to the leaching area. Pressure reducing valves are provided every 62 m down the shaft. The barren solution flow is then distributed to spray on the leaching stopes using Sennigner wobblers. The initial design just allows for 6 stopes to be started up in the first year.

The solution then percolates down the stopes and the pregnant solution leaving the bottom of the 6 stopes is collected in a sump. This solution is then pumped to the main sump for transfer to surface. Provision is also made to recirculate the pregnant solution through the stopes again.

The solution is pumped to the surface in 3 stages. Each stage consists of two 3-stage centrifugal pumps, each size to handle 50% of the pregnant solution flow rate.

6 - CAPITAL COST ESTIMATE
UNDERGROUND PUMPING & PIPING

CAPITAL COST ESTIMATE

UNDERGROUND PUMPING AND PIPING

Direct Costs	Can $
Equipment	233,000
Piping	475,000
Freight	15,000
Total Direct Costs	723,000
Indirect Costs	
Field indirect costs	21,000
Engineering and construction management	106,000
Construction insurance	8,500
Contingency	128,000
Total Indirect Costs	263,000
Total Costs	986,500

Notes

1. The capital cost estimates are of an order of magnitude type with an overall accuracy of +25%. All costs are expressed in Canadian dollars as of March 1985. Costs and prices received in US dollars have been converted to Canadian dollars at the rate of Can $1.40 per US dollar.

2. The estimate has been produced from a preliminary list of equipment and materials. Costs are allowed for procurement and installation of all piping materials and pumps. Foundations, base preparations, buildings, service connections for electrics, etc, was not included.

Volume

4

CENTRE DE RECHERCHE NORANDA

CENTRE DE RECHERCHE NORANDA

CONTRACT RESEARCH REPORT NO. 8515 CR 85-02

CONTRACT REPORT NO. OSQ84-00231

A PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT

DSS FILE NO. 07SQ. 23440-4-9071

MILESTONE 4 REPORT

OVERALL ASSESSMENT AND SENSITIVITY ANALYSIS

Project N-8515

April 1985

61 pp

MICROMEDIA

Department of Extractive Metallurgy
and Mining Technology Division
Centre de Recherche Noranda
240 boulevard Hymus
Pointe Claire, Québec
H9R 1G5

DECLASSIFIED
DATE 18/2/86
AUTHORIZED BY

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SUMMARY

The direct operating costs to produce copper by bacterially-assisted in-situ leaching of an underground low-grade copper sulphide deposit (0.9%), assuming a recovery of 55%, are estimated at $4.45/kg.

When considering the discounted cash flow at 15%, the assumed 55% recovery over a 10-year period is equivalent to a present recovery of only 28.9% which in turn is equivalent to present operating costs of $8.47/kg.

The state-of-the-art of bacterially-assisted leaching of sulphide copper is not yet sufficiently advanced to be economically feasible.

In order for bacterially-assisted in-situ leaching technology to become economically viable in the application studied, it will require drastic improvements particularly in the following aspects:

- faster rates of copper leaching
- a less expensive method of ore rubblization than drilling and blasting
- a reduction or elimination of the costs of pumping the dilute leach solution to surface.

SOMMAIRE

Les coûts direct d'opération pour produire du cuivre par lixiviation bactériologiquement assistée pour un dépôt de cuivre à basse teneur (0.9%), en assumant une récupération de 55%, sont estimés à $4.45/kg.

Lorsque l'on considère la valeur présente des revenues générés estimés à 15%, la récupération de 55% sur une période de 10 ans est équivalente à une récupération présente de 28.9%. Ceci est équivalent à des coûts actuel d'opération de $8.47/kg.

La présente technologie n'est pas encore assez avancée pour être rentable.

De façon à rendre la technologie de la lixiviation bactériologiquement assistée rentable dans la présente étude, d'important changements seraient requis et ce particulièrement dans les domaines suivants:

- augmentation de la cinétique d'extraction du suivre lors de la lixiviation
- un systeme moins dispendieux pour la fragmentation du minerai comparativement au forage et au dynamitage
- la réduction des coûts de pompage de la solution diluée de lixiviation vers la surface

PART I

(a) The project was completed on schedule.

(b) The project was completed on budget.

(c) The project was completed free of any areas of concern on which Crown assistance or guidance was required.

PART II

(a) This report constitutes the completion in whole of the contract (Serial No. OSQ84-00231) and Task 4, Overall Assessment and Sensitivity Analysis as required.

(b) No variation from the plan of work was made during completion of Task 4 nor the contract as a whole.

(c) No major items of equipment were purchased for the completion of Task 4 nor the contract as a whole.

(d) No trips were made during the completion of Task 4. Others have been previously outlined in earlier Milestone Reports.

PART III

A copy of the "Plan of Work and Costs" follows this page. It illustrates the time allowance allotted to each task and the completion of the project.

Supply and Services Canada / Approvisionnements et Services Canada

CONTRACT PLAN AND REPORT FORM – FORMULE DE PROJET DE CONTRAT ET DE RAPPORT

CONTRACT TITLE – TITRE DU CONTRAT 'PRE-FEASIBILITY ANALYSIS OF BACTERIALLY-ASSISTED UNDERGROUND LEACHING OF A CANADIAN COPPER SULPHIDE DEPOSIT'

CONTRACTOR – ENTREPRENEUR Centre de Recherche Noranda NORANDA INC.

07SQ 23440-4-9071

	1984						1985					
	1. July	2 Aug.	3 Sept.	4 Oct.	5 Nov.	6 Dec.	7 Jan.	8 Feb.	9 Mar.	10 Apr	11 May	12
1. Mine Site Selection			━	━	━							
2. Pre-feasibility of Underground Operations					━	━	━	━	━			
3. Pre-feasibility of Met. Circuit					━	━	━	━	━			
4. Sensitivity Analysis									━			
			$20,000	$20,000	$15,000	$15,000	$15,000	$10,000	$ 5,000			

J. Martel

Table I - TIME/COST CHART

1. INTRODUCTION

The pre-feasibility study on bacterially-aided underground leaching of copper sulfides was divided into four phases.

In Phase I, all Noranda's Canadian copper mines were evaluated to establish the best one to use for the study and the Geco mine in Manitouwadge, Ontario was chosen based upon the following criteria: mine configuration, geology and metallurgical characteristics.

Phase II contains the design of the mining developments required before leaching. This includes the access to prepare the stopes, rubblization of ore in the stopes by drilling and blasting and excavations needed to collect the leach solution. A unitary cost of $14.74/ tonne of leachable ore, was estimated for the mining activity. This value is applicable within the range of copper production from 500 to 4000 tonnes copper/year considered in this study and was based upon the assumption that development ore could be milled on-site and that the revenues generated by this could be used as credit to the mining costs.

The leaching circuit and the recovery plant were designed and costed in Phase III. On the basis of lower capital costs and greater flexibility for adapting the recovery plant to in-situ leaching, particularly in the opening and closing years of the project, it was decided to use cementation as the recovery process instead of solvent extraction-electrowinning. The design capacity of the base case was fixed at 1000 tonnes Cu/year, from pregnant leach solution pumped to surface with 1 gpL Cu. The initial capital investment for the leaching and recovery circuits was estimated to be $3.4 million and the operating costs of producing refined copper was $1.47/kg (66.8¢/lb).

In this final Phase IV, "Overall Assessment and Sensitivity Analysis", the costs developed for the mining development and the metallurgical circuits have been used to establish the cash flows of the project for variations in:

- price of copper
- operating costs
- leaching rate and overall recovery
- grade of ore

Cummulative net present values (NPV's) have been estimated using a 15% discount rate and a comparison between these values was then used in the sensitivity analysis. Cash flow schedules spanned over a 19-year period (total time required to leach the ore) in constant 1985 Canadian dollars. Taxation was estimated at 40% of positive cash flows.

Two other case studies are presented to help clarify some of the more striking conclusions made in the study of bacterially-assisted underground leaching.

In Case 2 it was assumed that the ore grade was 1.8% Cu, double the value used in the base case. This results in a 50% reduction in mining development costs. In Case 3, in addition to improving the ore to 1.8% Cu , it was assumed that the (i) project could obtain 75% overall copper recovery, (ii) pregnant leach solutions could be concentrated to 2 gpL Cu (instead of 1 gpL used in the base case), so that the recovery plant could double its capacity for a slight increase in the capital cost, and (iii) unit mining costs could be reduced by ∿30% using larger drilling equipment. These two cases (particularly the latter) should be regarded as very optimistic but will highlight the cost of this technology even under the best circumstances.

Conventional mining costs and cash flow schedules were also calculated so that a comparison between the two technologies could be made.

Both leaching and conventional mining calculations have been based on the fact that the zone to be mined is a part of an operating mine and consequently, milling facilities exist. This serves to reduce the mining development costs in the leaching case and eliminates the need for any capital expenditures in the conventional mining case.

2. ECONOMICS OF BASE CASE STUDY

The main variables assumed for the base case study are:

- Leachable ore	2.1 million tonnes
- Ore grade	0.9% Cu
- Total copper recovery	55% (10,356 mt)
- Rate of copper extraction	10% first year 1% less each subsequent year
- Total leaching time	18 years
- Pregnant leach liquor concentration	1 gpL Cu
- Designed annual copper production	1000 mt
- Mine development cost	$14.74/tonne
- Concentrate transportation	$38.58/tonne
- Smelter deductions	11 kg*/tonne of concentrate
- Smelting and refining charges for cement copper	$0.29/kg
- Leaching and recovery plant capital (cementation route)	$3.37 M
- Leaching and recovery plant operating cost (cementation route)	$1.13/kg Cu (51.5¢/lb Cu)
- Price of copper	$2.20/kg Cu ($1/lb Cu)

Under these base case conditions, the NPV was estimated to be -$22.1 million. A zero NPV (which implies a 15% ROI) is only obtained by:

- reducing the overall operating costs by 87% to 19¢/kg Cu (8.6¢/lb Cu) (Figure 1)
- increasing the selling price of copper to $9.22/kg Cu ($4.18/lb Cu) (Figure 2)

This clearly indicates that bacterially-assisted in-situ leaching of chalcopyrite is not economically viable under the base case conditions.

The effect of the mining costs on the NPV was also examined and is represented in Figure 3. The analysis shows that even if the mining costs were reduced to zero, the NPV of the base case is still negative (-$0.5 million), indicating that at current copper prices not even the leaching and recovery circuits can be operated profitably for an in-situ bacterial leach project.

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*cost is dependant on smelter tolling charges

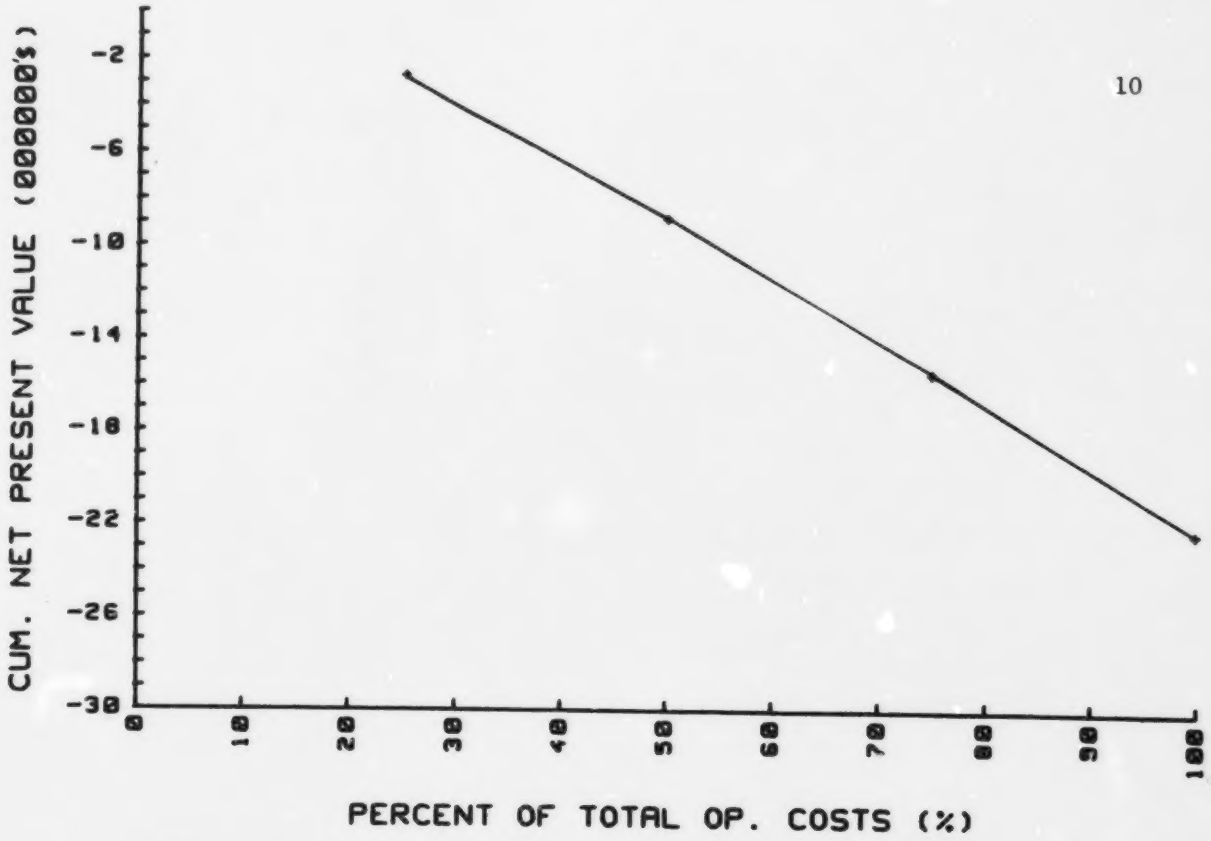

Figure 1 - Base Case Sensitivity Analysis : Total Operating Costs

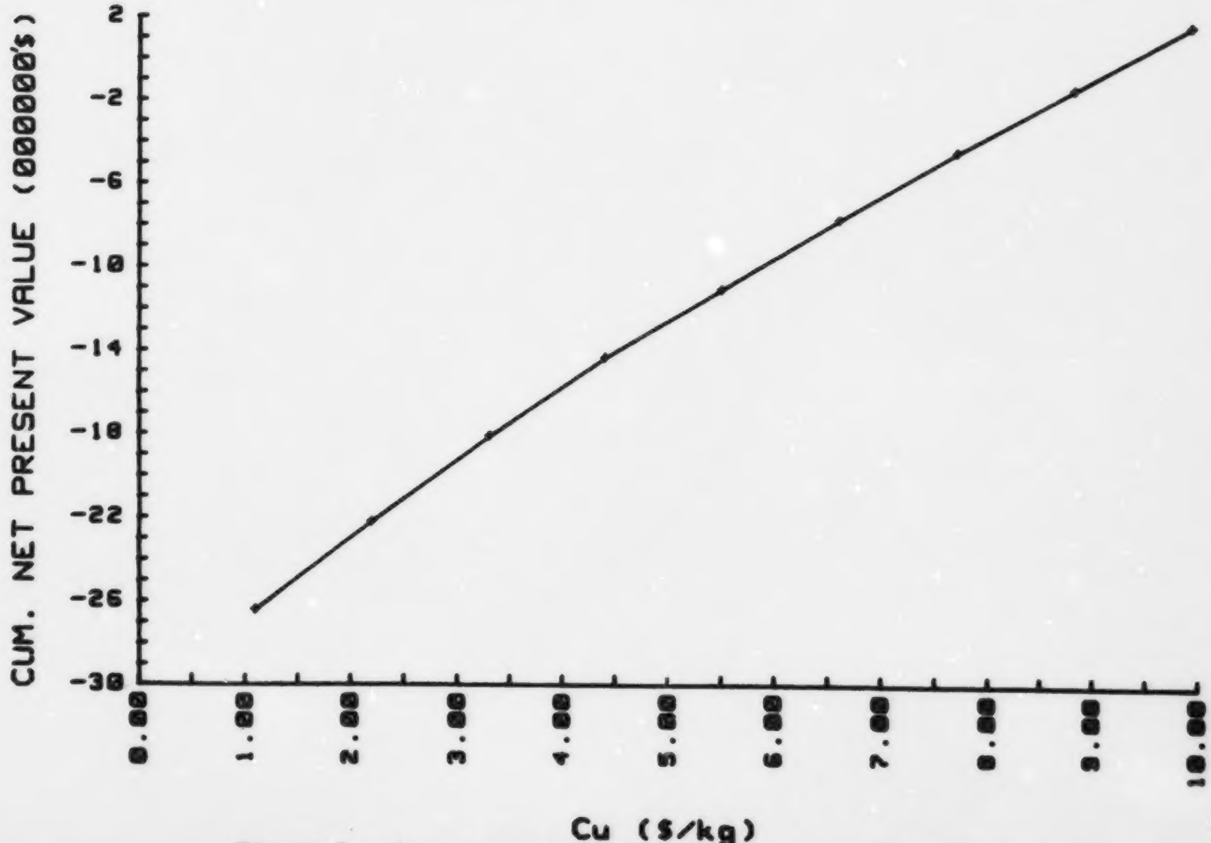

Figure 2 - Base Case Sensitivity Analysis : Copper Price

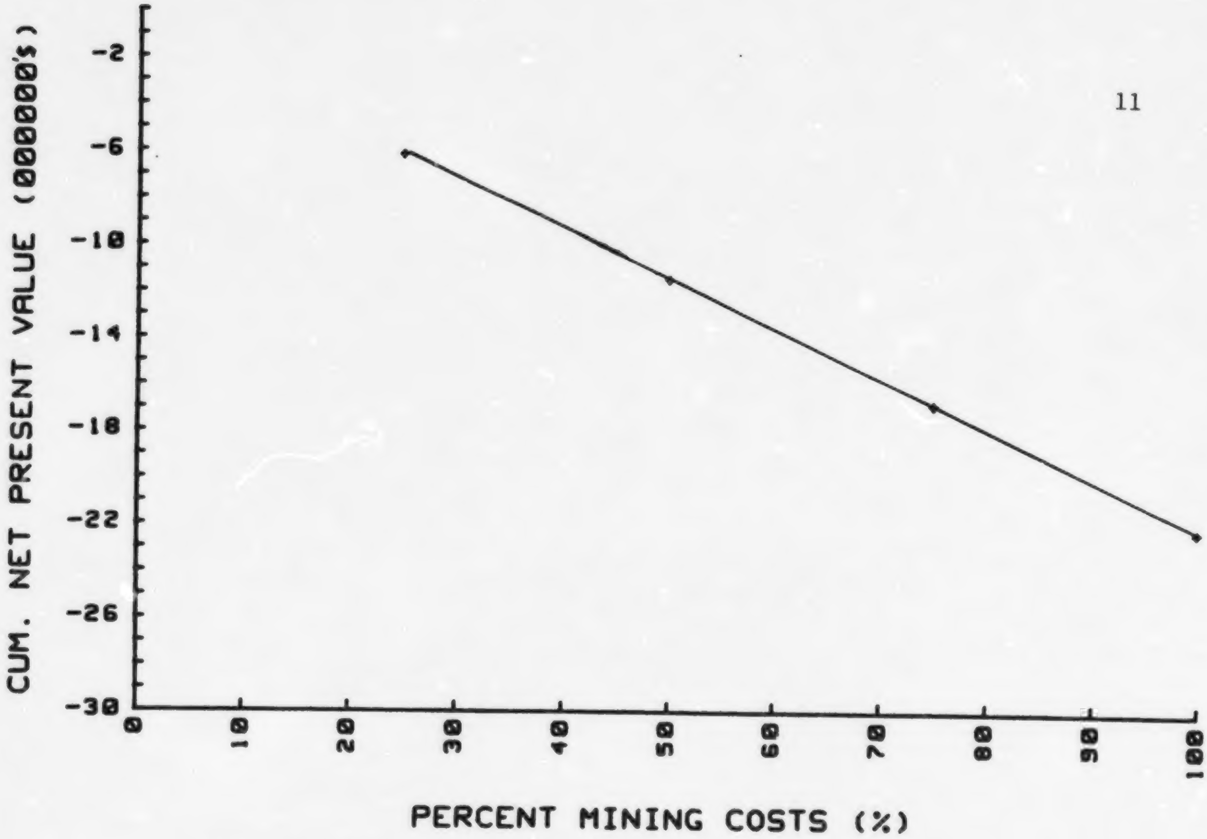

Figure 3 - Base Case Sensitivity Analysis : Mining Costs

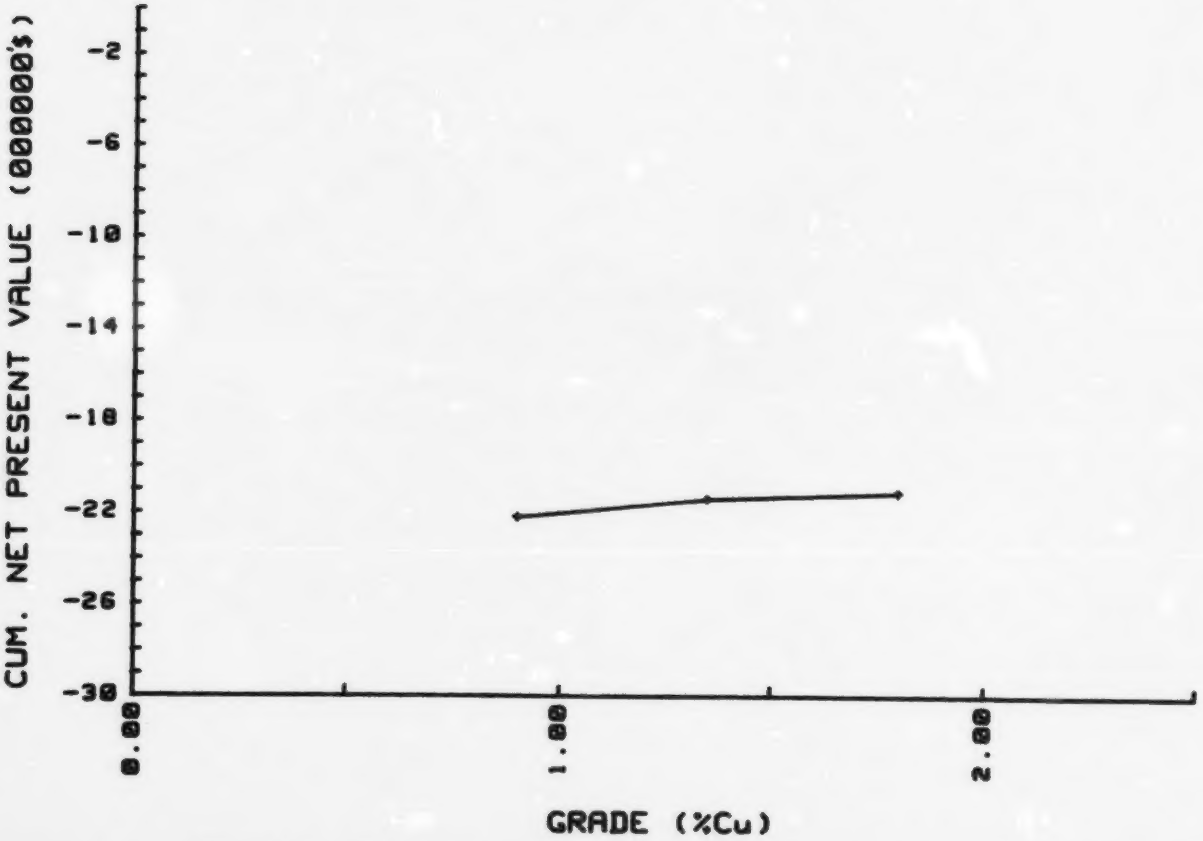

Figure 4 - Base Case Sensitivity Analysis : Copper Grade

The effects of ore grade (0.9 to 1.8% Cu) and overall copper recoveries (55-100%) at constant leaching rates on the NPV are represented in Figures 4 and 5, respectively. Contrary to what would be expected, these graphs show that neither of these variable significantly affects the economics of in-situ bacterial leaching. Based upon these analyses, it can be concluded that the most significant variable affecting the economic viability of this project is the slow (and incomplete) copper extraction which results in slow realization of revenues. In effect, if the cash flow is discounted as in this study, the 55% copper recovery is equivalent to a present recovery of only 28.9%.

3. CASE STUDIES

The negligible effect of higher ore grade and overall recoveries on the NPV can be further exemplified from the following cases:

3.1 CASE 2 Study

The main variables differing from the base case study (section 2) are:

• ore grade	1.8% Cu
• leachable ore	1.1 million tonnes

All other variables were kept constant.

The main advantage of this over the base case is that the number of stopes required for leaching are reduced by 50% and thus mining development costs are also reduced by 50%. The cash flow schedule is shown in Appendix B.

The NPV for this case was estimated to be -$11.5 million. A zero NPV is obtained by increasing the selling price of copper to $6/kg ($2.72/lb).

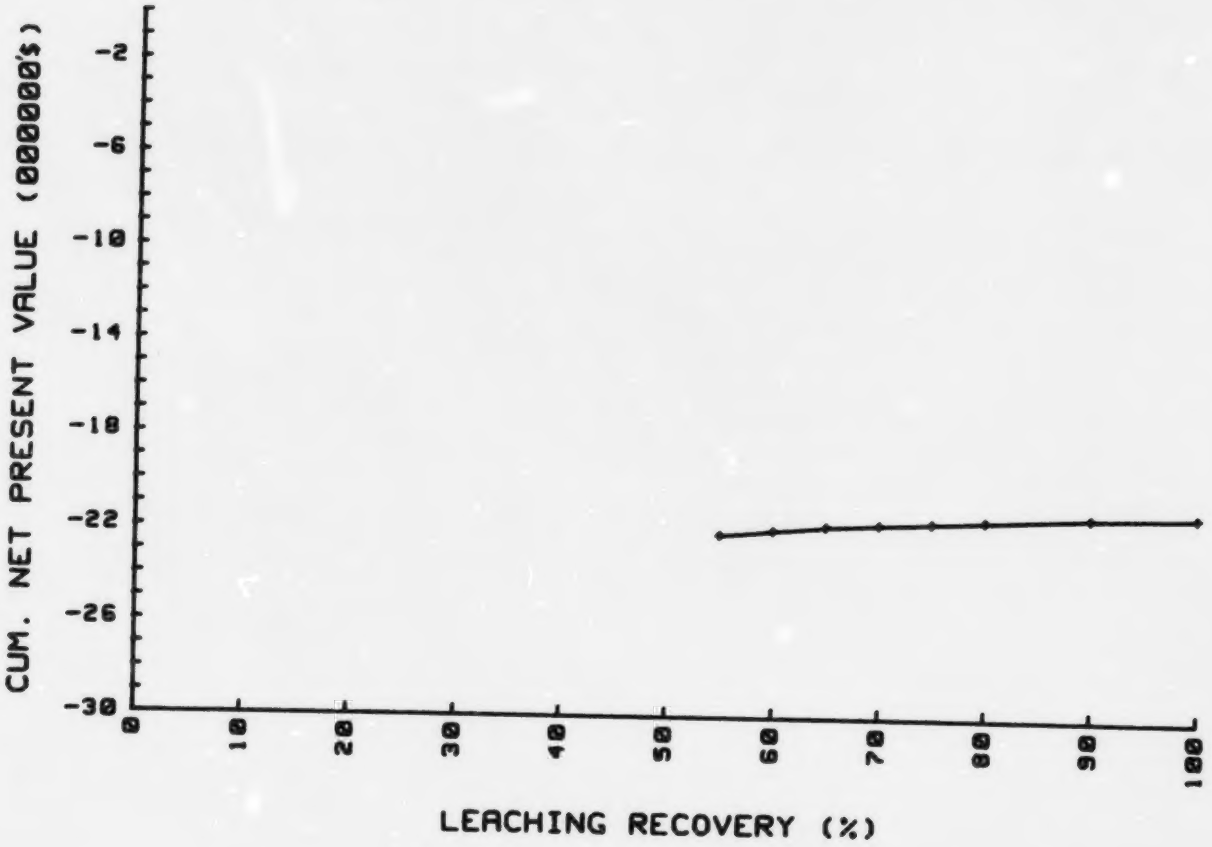

Figure 5 : Base Case Sensitivity Analysis : Leaching Recovery

3.2 CASE 3 Study

The main variables differing from the base case study are:

• ore grade	1.8% Cu
• leachable ore	1.1 million tonnes
• total copper recovery	75%
• pregnant leach liquor concentration	2 gpL Cu
• mine development cost	$10.58/tonne
• leach/recovery operating cost	$1.20/kg Cu (54.4¢/lb Cu)

The reduction of mine development costs from $14.74 to $10.57/tonne was based upon the use of 152 mm drill holes instead of 114 mm drill holes for the base case.

The main advantages for this case over the base case study are:

- the unitary mining development costs are reduced by 27%
- the overall mining development costs are reduced by 62%
- the leaching circuit operating costs are reduced by pumping a more concentrated solution to surface
- the rate of annual copper recovery and overall extraction are increased over the 10-year period.

The NPV for this case was estimated to be -$8.1 million (Appendix C). Even under these highly (or over) optimistic conditions, the price of copper would have to be $3.63/kg ($1.65/lb) in order for this project to be viable.

4. CONVENTIONAL MINING

The economics of producing copper by conventional mining and milling was also examined. For this study, it was decided to use the same ore grade and production tonnage as for the base case. As is the case at Geco, it was assumed that the project does not require any capital and that the existing mill can process the ore hauled to surface.

Unitary mining development costs of \$17.76/tonne were used for conventional mining instead of \$14.74/tonne used for the in-situ leach project. This difference is largely due to the credit from the copper recovered in the mill from development ore for the in-situ leach project (removed to allow for swelling). Costs for milling, smelting, refining and transportation are the same as used in Phase II and in the base case of this report.

It was estimated that the 2.1 million tonnes of ore could be mined conventionally at Geco within 6 years so that the cashflow schedule for this case only spans 6 years.

Figure 6 shows that this case generates an NPV of -\$25.3 million; only -\$3.2 million less than the base case for bacterial leaching. A zero NPV is obtained however by:

- reducing the overall operating costs by 45% (Figure 6) , or
- increasing the selling price of copper to \$3.94/kg Cu (\$1.8/lb Cu) (Figure 7)

The effect of ore grade on conventional mining NPV's was also examined and is shown in Figure 8. It can be seen that, unlike bacterial in-situ leaching, ore grade has a significant effect on the economics of the project, reaching an NPV of zero at an ore grade of only 1.9% Cu.

5. COMPARISON OF THE BASE CASE AND CONVENTIONAL MINING

Figures 9 to 11 show that conventional mining is much more sensitive than in-situ bacterial leaching to variations in copper price, operating costs and grade. Table II summarizes the conditions required for both techniques to generate a zero NPV and thus a 15% ROI.

Although neither conventional mining nor in-situ bacterial leaching of low-grade sulphide ores are economically attractive under present economic conditions, the former requires lower copper prices and ore grades (Figure 9) to generate a zero NPV. It should be

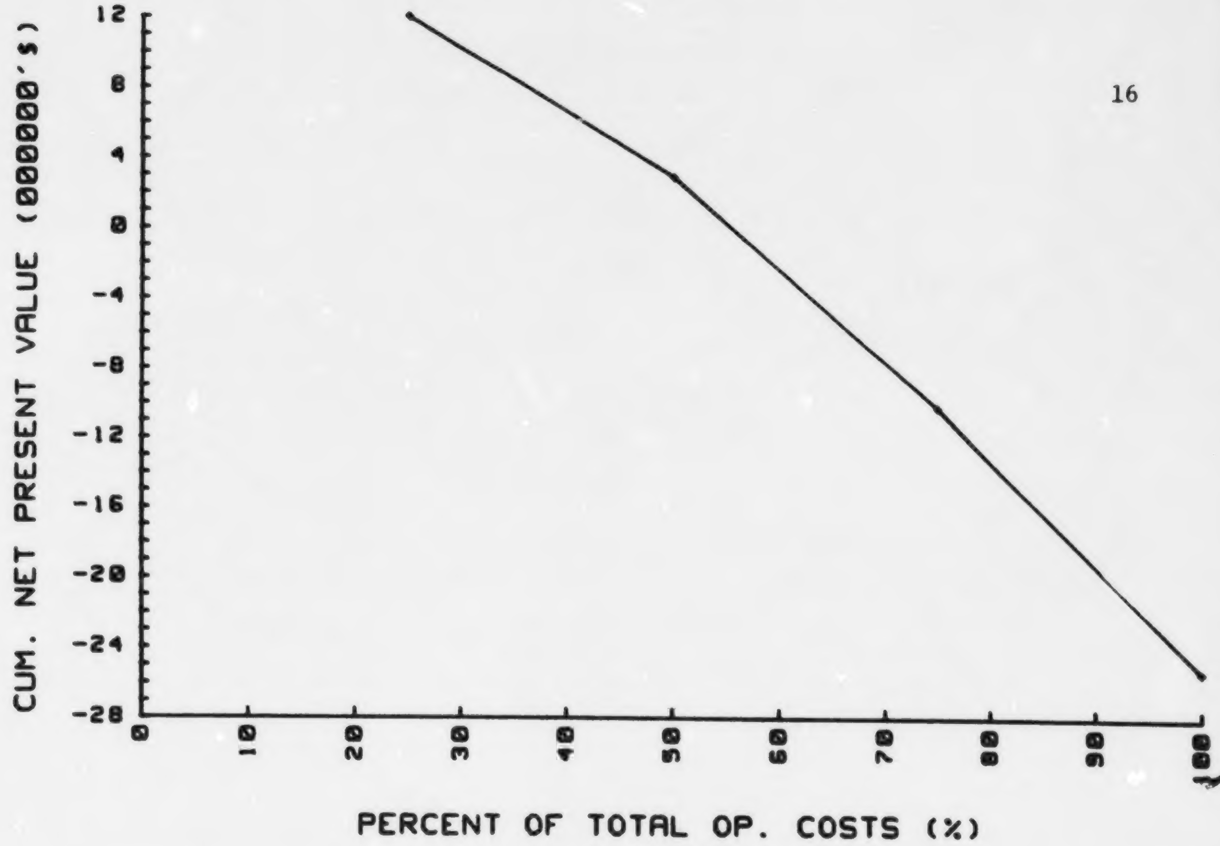

Figure 6 - Conventional Mining Sensitivity Analysis : Total Operating Costs

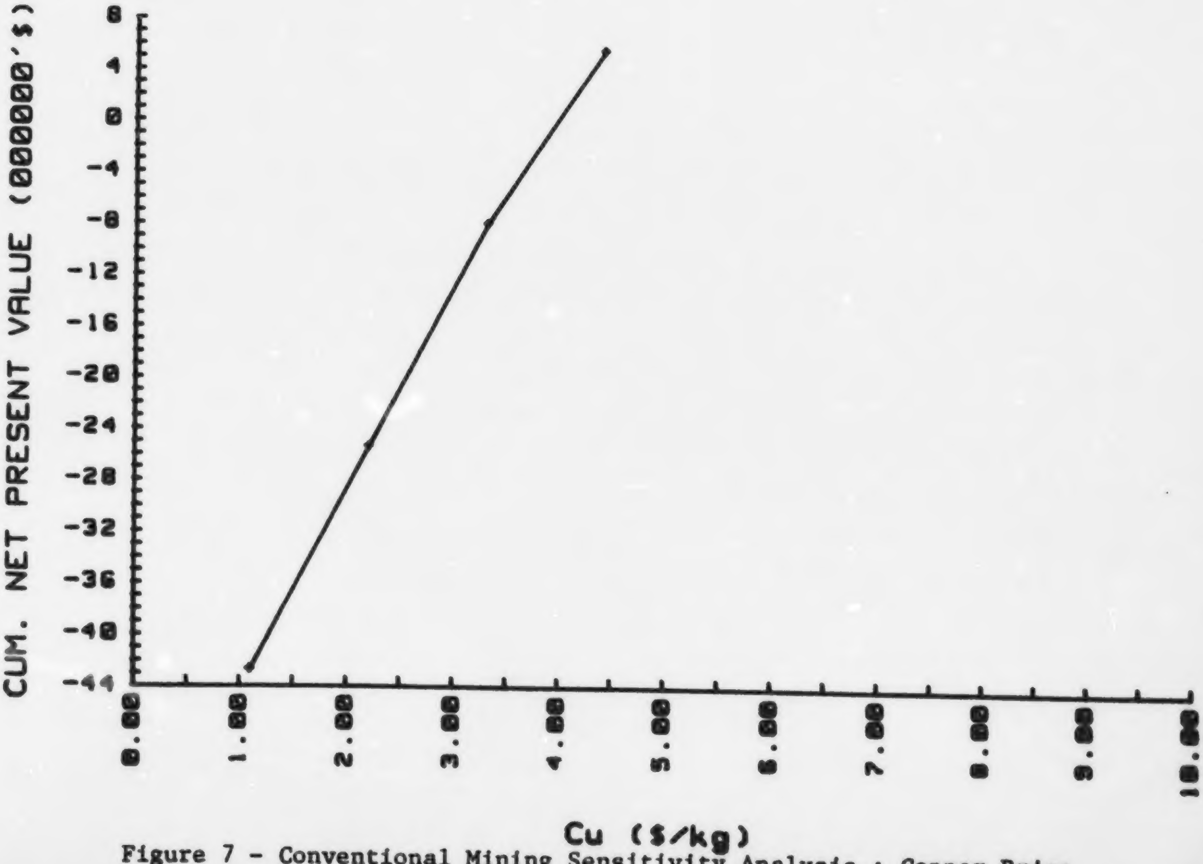

Figure 7 - Conventional Mining Sensitivity Analysis : Copper Price

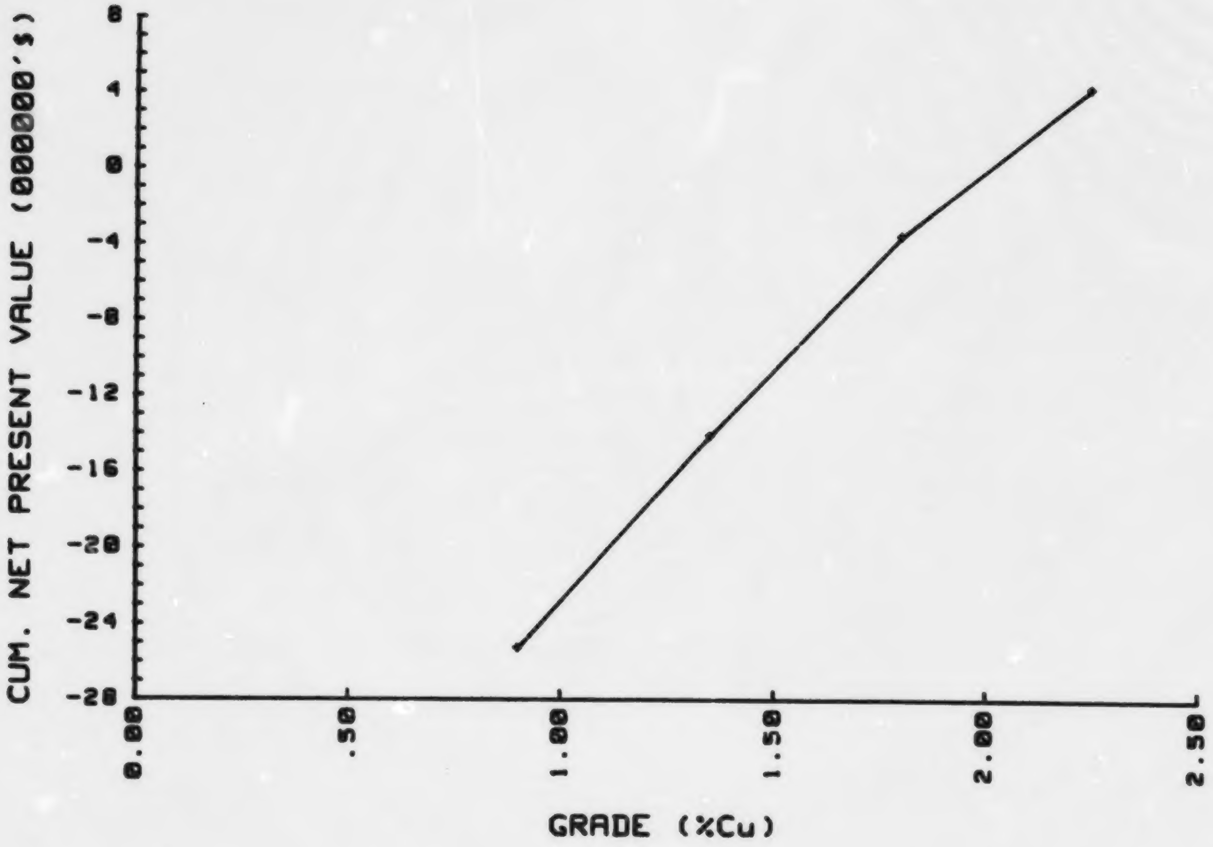

Figure 8 - Conventional Mining Sensitivity Analysis : Copper Grade

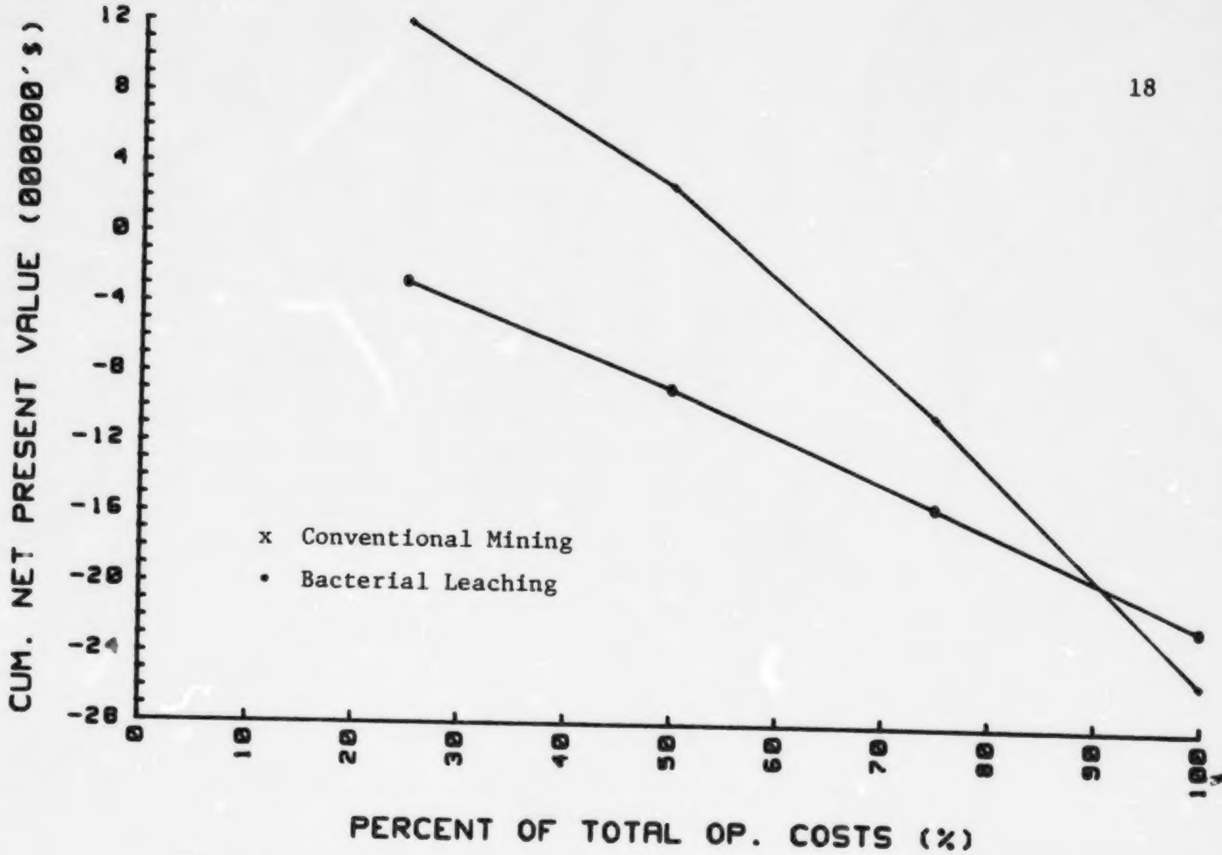

Figure 9 - Comparative Sensitivity Analysis : Total Operating Costs

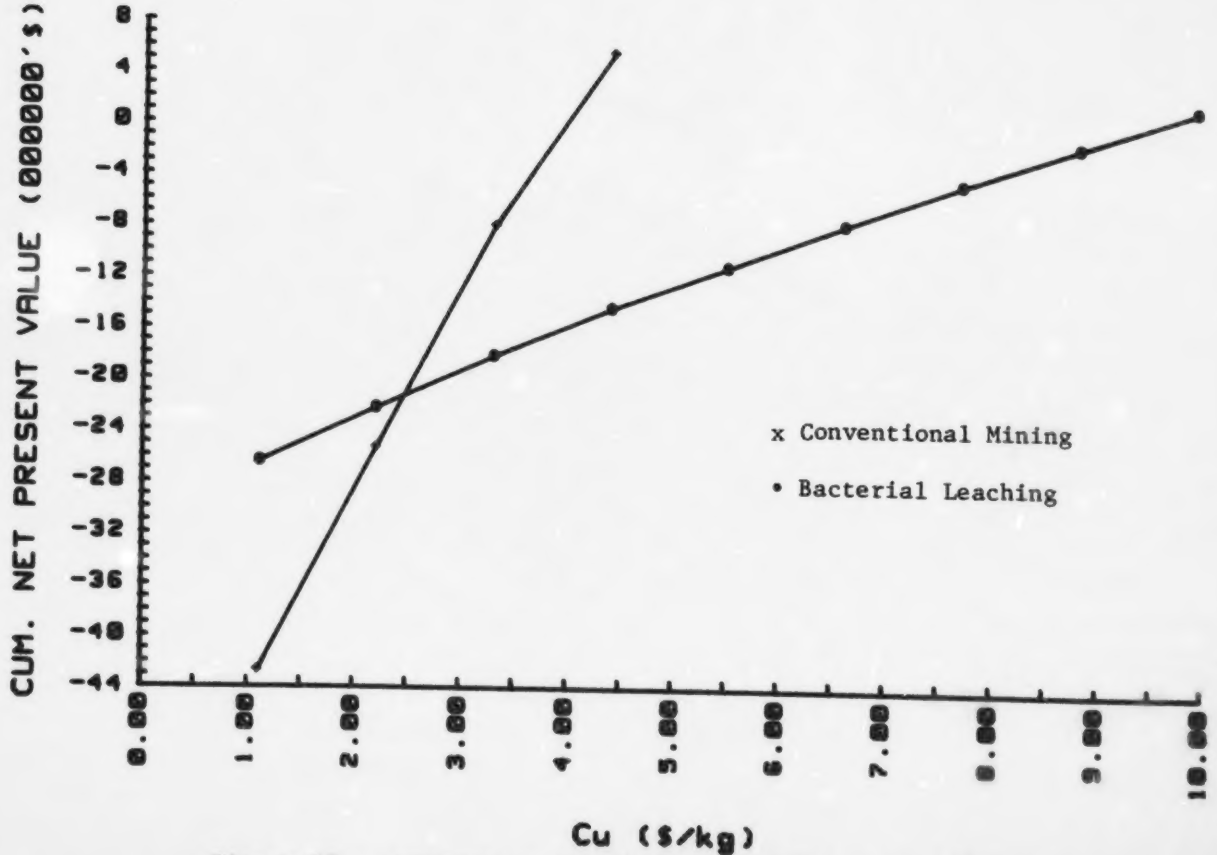

Figure 10 - Comparative Sensitivity Analysis : Copper Price

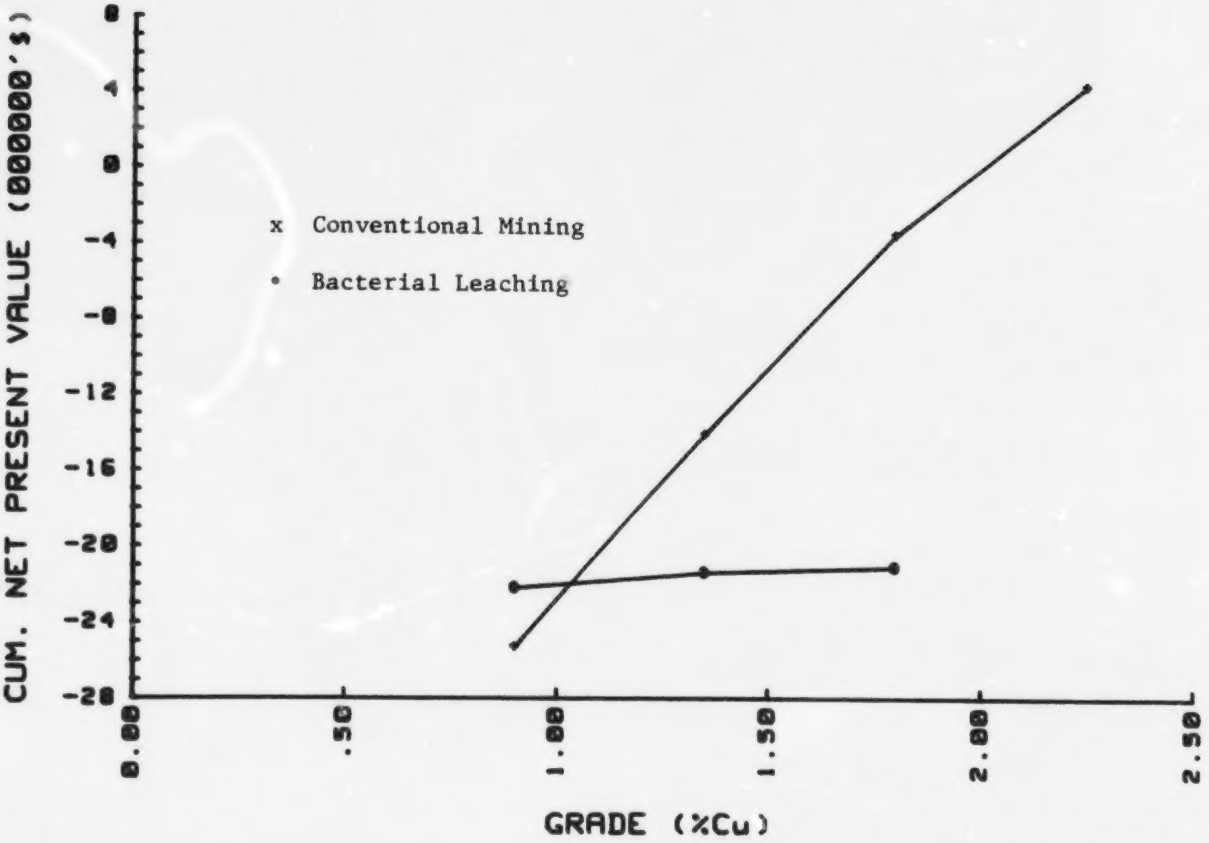

Figure 11 - Comparative Sensitivity Analysis : Copper Grade

TABLE II

CONDITIONS REQUIRED FOR CONVENTIONAL MINING AND IN-SITU BACTERIAL LEACHING TO GENERATE A ZERO NPV[1]

Conventional Mining	In-Situ Bacterial Leaching
• at an ore grade of 0.9% Cu, a selling price of $3.94/kg Cu is required	• at an ore grade of 0.9% Cu, a selling price of $9.22/kg Cu is required
• at a Cu selling price of $2.20/kg minimum ore grade required is 1.9% Cu	• at a Cu selling price of $2.20/kg Cu, ore grade does not have a significant effect on NPV's within the limits investigated
• at a copper ore grade of 0.9% and selling price of $2.20/kg Cu, total operating costs should be reduced by 45%	• at a copper ore grade of 0.9% Cu and selling price of $2.20/kg Cu, the total operating costs (mining, leaching and recovery) should be reduced by 87%

[1] A zero NPV requires 15% ROI (see text)

stressed again that this comparison is only valid when a mine and mill exist, and have been amortized.

It is difficult to estimate what potential exists for reducing the operating costs of either conventional mining or in-situ bacterial leaching to a point that could make each alternative economically viable. It could be argued that it is more likely that greater cost reduction will be obtained in an emerging technology such as in-situ bacterial leaching than in one that has existed for years. However, since up to 66% of the operating costs for in-situ bacterial leaching are due to the mining development needed for adequate rubblization of ore, it should be expected that major improvements in in-situ bacterial leaching resulting from more economic rubblization methods, may also reduce the costs of conventional mining.

Only if the leaching aspects are improved significantly, particularly the rate of copper extraction, could in-situ bacterial leaching technology become more attractive than conventional mining technology.

6. CONCLUSION

Bacterially-assisted in-situ leaching of low-grade chalcopyrite ore is not economically viable under present economic conditions. The sensitivity study has shown that improved overall recovery (up to 100%) and better ore grade (up to 1.8% Cu) have little impact on the process economics and an 87% reduction of all operating costs (mining development, leaching and the recovery plant) would be required in order to produce a zero NPV discounted at 15%/year at a copper price of $2.20/kg.

The major factor that affects in-situ bacterial leaching technology is the slow realization of revenues caused by the slow leaching of copper. When considering the time value of money, a 55% copper recovery over a 10-year leaching period is equivalent to a present recovery of only 29%.

Although conventional mining was also found not to be economically viable under the conditions studied, the sensitivity analysis showed that higher copper prices and better ore grade had a greater impact on the process economics than in the case of in-situ bacterial leaching. The difference between the two technologies is due primarily to the fact that mining development costs in in-situ bacterial leaching to rubblize the ore before leaching could only be reduced to 80% of the costs of conventional mining and milling where revenues from the recovery of 80% of the copper in the zone mined would be obtained within a 3-month period from the time in which the costs are incurred. On the other hand, in-situ bacterial leaching produces only 10% of the copper in the area being leached in the first years.

In order for in-situ bacterial leaching technology to become economically viable in the application studied, it will require drastic improvements, particularly in the following aspects:

- faster rates of copper leaching
- a less expensive system of ore rubblization than drilling and blasting
- a reduction or elimination of the costs of pumping the dilute leach solution to surface.

APPENDIX A

BASE CASE DATA

	YEAR																		
STATISTICS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.89	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.08	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.08	-5091.30	-4869.38	-4670.82	712.40	-1856.52	-1891.50	-1872.88	-1865.87	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-5091.30	-4869.38	-4670.82	712.40	-1856.52	-1891.50	-1872.88	-1865.87	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	284.96	0.00	0.00	0.00	0.00	292.47	261.63	205.57	154.18	112.13	79.42	56.06	37.38	21.02	11.68
AFTER TAX CASH FLOW (000'$):	-8576.08	-5091.30	-4869.38	-4670.82	427.44	-1856.52	-1891.50	-1872.88	-1865.87	438.70	392.45	308.35	231.27	168.19	119.14	84.10	56.07	31.54	17.52
CUM. NET PRESENT VALUE (000'$):	-8576	-13003	-16685	-19756	-19512	-20435	-21253	-21957	-22567	-22442	-22345	-22279	-22236	-22208	-22191	-22181	-22175	-22172	-22171
CUM. INTERNAL RATE OF RETURN (%) :					-0.92	-0.55	-0.47	-0.41	-0.36	-0.81	-0.58	-0.46	-0.40	-0.36	-0.33	-0.31	-0.29	-0.29	-0.28

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 2. BASE CASE SCHEDULE

	YEAR 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	350.43	665.82	946.17	1191.26	1051.51	1086.34	1107.37	1107.37	1086.34	876.08	665.82	490.61	350.43	245.30	175.22	105.13	52.56	17.52
PROD. REVENUE: (000'$)	0.00	350.43	665.82	946.17	1191.26	1051.51	1086.34	1107.37	1107.37	1086.34	876.08	665.82	490.61	350.43	245.30	175.22	105.13	52.56	17.52
CASH REVENUE: (000'$)	0.00	292.03	613.26	899.44	1150.41	1074.80	1080.54	1103.86	1107.37	1089.85	911.13	700.87	519.81	373.79	262.82	186.90	116.81	61.33	26.28
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.08	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.08	-5383.32	-5482.63	-5570.26	-438.02	-2931.32	-2972.03	-2976.74	-2973.24	-358.68	-257.04	-186.94	-134.37	-93.47	-64.26	-46.74	-23.37	-8.76	2.92
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-5383.32	-5482.63	-5570.26	-438.02	-2931.32	-2972.03	-2976.74	-2973.24	-358.68	-257.04	-186.94	-134.37	-93.47	-64.26	-46.74	-23.37	-8.76	2.92
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	1.17
AFTER TAX CASH FLOW (000'$):	-8576.08	-5383.32	-5482.63	-5570.26	-438.02	-2931.32	-2972.03	-2976.74	-2973.24	-358.68	-257.04	-186.94	-134.37	-93.47	-64.26	-46.74	-23.37	-8.76	1.75
CUM. NET PRESENT VALUE (000'$):	-8576	-13257	-17403	-21065	-21316	-22773	-24058	-25177	-26149	-26251	-26315	-26355	-26380	-26395	-26404	-26410	-26412	-26413	-26413
CUM. INTERNAL RATE OF RETURN (%) :																				-0.86

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 3. BASE CASE WITH A COPPER PRICE OF $1.10/kg

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	1051.30	1997.47	2838.50	3573.79	3154.52	3259.02	3322.10	3322.10	3259.02	2628.25	1997.47	1471.82	1051.30	735.91	525.65	315.39	157.69	52.56
PROD. REVENUE: (000'$)	0.00	1051.30	1997.47	2838.50	3573.79	3154.52	3259.02	3322.10	3322.10	3259.02	2628.25	1997.47	1471.82	1051.30	735.91	525.65	315.39	157.69	52.56
CASH REVENUE: (000'$)	0.00	876.08	1839.77	2698.33	3451.24	3224.40	3241.61	3311.59	3322.10	3269.56	2733.38	2102.60	1559.43	1121.38	788.47	560.69	350.43	183.98	78.85
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.08	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.08	-4799.27	-4256.12	-3771.37	1862.81	-781.72	-810.96	-769.02	-758.50	1821.01	1565.21	1214.79	905.25	654.12	461.39	327.06	210.25	113.89	55.48
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-4799.27	-4256.12	-3771.37	1862.81	-781.72	-810.96	-769.02	-758.50	1821.01	1565.21	1214.79	905.25	654.12	461.39	327.06	210.25	113.89	55.48
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	745.12	0.00	0.00	0.00	0.00	728.40	626.08	485.92	362.10	261.65	184.55	130.82	84.10	45.55	22.19
AFTER TAX CASH FLOW (000'$):	-8576.08	-4799.27	-4256.12	-3771.37	1117.69	-781.72	-810.96	-769.02	-758.50	1092.61	939.12	728.87	543.15	392.47	276.83	196.24	126.15	68.33	33.29
CUM. NET PRESENT VALUE (000'$):	-8576	-12749	-15968	-18447	-17808	-18197	-18548	-18837	-19095	-18774	-18542	-18385	-18284	-18220	-18181	-18157	-18143	-18137	-18134
CUM. INTERNAL RATE OF RETURN (%) :					-0.78	-0.55	-0.47	-0.41	-0.36	-0.44	-0.31	-0.25	-0.22	-0.20	-0.18	-0.17	-0.17	-0.16	-0.16

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 4. BASE CASE WITH A COPPER PRICE OF $3.31/kg Cu

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	1401.73	2663.29	3784.67	4765.05	4206.03	4345.37	4429.47	4429.47	4345.37	3504.33	2663.29	1962.42	1401.73	981.21	700.87	420.52	210.26	70.09
PROD. REVENUE: (000'$)	0.00	1401.73	2663.29	3784.67	4765.05	4206.03	4345.37	4429.47	4429.47	4345.37	3504.33	2663.29	1962.42	1401.73	981.21	700.87	420.52	210.26	70.09
CASH REVENUE: (000'$)	0.00	1168.11	2453.03	3597.78	4601.65	4299.20	4322.14	4415.45	4429.47	4359.38	3644.50	2803.46	2079.23	1495.18	1051.30	747.59	467.24	245.30	105.13
OPERATING COSTS (000'$)																			
MINE:	5208.00	5208.00	5208.00	5208.00	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.00	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.00	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.00	-4507.24	-3642.86	-2871.93	3013.22	293.08	269.57	334.85	348.86	2910.86	2476.33	1915.65	1425.06	1027.91	724.21	513.96	327.06	175.21	81.77
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.00	-4507.24	-3642.86	-2871.93	3013.22	293.08	269.57	334.85	348.86	2910.86	2476.33	1915.65	1425.06	1027.91	724.21	513.96	327.06	175.21	81.77
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	1205.29	117.23	107.83	133.94	139.55	1164.34	990.53	766.26	570.02	411.17	289.68	205.58	130.83	70.09	32.71
AFTER TAX CASH FLOW (000'$):	-8576.00	-4507.24	-3642.86	-2871.93	1807.93	175.85	161.74	200.91	209.32	1746.51	1485.80	1149.39	855.04	616.75	434.53	308.37	196.24	105.13	49.06
CUM. NET PRESENT VALUE (000'$):	-8576	-12495	-15250	-17138	-16105	-16017	-15947	-15872	-15803	-15307	-14940	-14683	-14533	-14432	-14371	-14333	-14312	-14302	-14298
CUM. INTERNAL RATE OF RETURN (%) :					-0.65	-0.60	-0.54	-0.46	-0.40	-0.23	-0.17	-0.13	-0.11	-0.10	-0.09	-0.09	-0.08	-0.08	-0.08

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 5. BASE CASE WITH A COPPER PRICE OF $4.41/kg

STATISTICS	YEAR 1	2	3	4	5	6	7	8	9	10
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986
Recirculated(t):	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986
SALES:	0.00	1752.16	3329.11	4730.84	5956.31	5257.54	5431.71	5536.84	5536.84	5431.71
PROD. REVENUE: (000'$)	0.00	1752.16	3329.11	4730.84	5956.31	5257.54	5431.71	5536.84	5536.84	5431.71
CASH REVENUE: (000'$)	0.00	1460.14	3066.29	4497.22	5752.07	5374.00	5402.68	5519.31	5536.84	5449.23
OPERATING COSTS (000'$)										
MINE:	5208.00	5208.00	5208.00	5208.08	0.00	2604.04	2604.04	2604.04	2604.04	
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53
TOTAL:	5208.00	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53
OPERATING PROFIT :	-5208.08	-4215.21	-3029.61	-1972.49	4163.64	1367.88	1350.11	1438.71	1456.23	4000.70
CAPITAL COSTS (000'$)										
MINE:										
LEACH:	1636.00									
PLANT:	1732.00									
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-4215.21	-3029.61	-1972.49	4163.64	1367.88	1350.11	1438.71	1456.23	4000.70
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	1665.45	547.15	540.04	575.48	582.49	1600.28
AFTER TAX CASH FLOW (000'$):	-8576.08	-4215.21	-3029.61	-1972.49	2498.18	820.73	810.07	863.23	873.74	2400.42
CUM. NET PRESENT VALUE (000'$):	-8576	-12241	-14532	-15829	-14401	-13993	-13643	-13318	-13032	-12350
CUM. INTERNAL RATE OF RETURN (%) :					-0.53	-0.42	-0.34	-0.26	-0.21	-0.12

STATISTICS	11	12	13	14	15	16	17	18	19
Cu in Soln.(t):	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0
Produced Cu (t):	795	604	445	318	223	159	95	48	16
SALES:	4380.41	3329.11	2453.03	1752.16	1226.51	876.08	525.65	262.82	87.61
PROD. REVENUE: (000'$)	4380.41	3329.11	2453.03	1752.16	1226.51	876.08	525.65	262.82	87.61
CASH REVENUE: (000'$)	4555.63	3504.33	2599.04	1868.97	1314.12	934.49	584.05	306.63	131.41
OPERATING COSTS (000'$)									
MINE:									
PROCESSING:	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	3387.46	2616.52	1944.87	1401.71	987.04	700.85	443.87	236.54	108.05
CAPITAL COSTS (000'$)									
MINE:									
LEACH:									
PLANT:									
TOTAL:	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	3387.46	2616.52	1944.87	1401.71	987.04	700.85	443.87	236.54	108.05
TAXES PAYABLE @ 40% (000'$):	1354.98	1046.61	777.95	560.68	394.81	280.34	177.55	94.62	43.22
AFTER TAX CASH FLOW (000'$):	2032.47	1569.91	1166.92	841.02	592.22	420.51	266.32	141.92	64.83
CUM. NET PRESENT VALUE (000'$):	-11848	-11510	-11292	-11156	-11072	-11020	-10992	-10978	-10973
CUM. INTERNAL RATE OF RETURN (%) :	-0.08	-0.06	-0.04	-0.03	-0.03	-0.02	-0.02	-0.02	-0.02

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 6. BASE CASE WITH A COPPER PRICE OF $5.51/kg

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	2102.60	3994.93	5677.01	7147.57	6309.04	6518.05	6644.20	6644.20	6518.05	5256.49	3994.93	2943.63	2102.60	1471.82	1051.30	630.78	315.39	105.13
PROD. REVENUE: (000'$)	0.00	2102.60	3994.93	5677.01	7147.57	6309.04	6518.05	6644.20	6644.20	6518.05	5256.49	3994.93	2943.63	2102.60	1471.82	1051.30	630.78	315.39	105.13
CASH REVENUE: (000'$)	0.00	1752.16	3679.54	5396.66	6902.48	6448.80	6483.21	6623.18	6644.20	6539.07	5466.75	4205.19	3118.85	2242.77	1576.95	1121.38	700.87	367.95	157.69
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.08	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.08	-3923.19	-2416.35	-1073.04	5314.05	2442.68	2430.64	2542.57	2563.60	5090.55	4298.58	3317.39	2464.68	1775.50	1249.86	887.75	560.69	297.86	134.33
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-3923.19	-2416.35	-1073.04	5314.05	2442.68	2430.64	2542.57	2563.60	5090.55	4298.58	3317.39	2464.68	1775.50	1249.86	887.75	560.69	297.86	134.33
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	2125.62	977.07	972.26	1017.03	1025.44	2036.22	1719.43	1326.95	985.87	710.20	499.94	355.10	224.27	119.15	53.73
AFTER TAX CASH FLOW (000'$):	-8576.08	-3923.19	-2416.35	-1073.04	3188.43	1465.61	1458.39	1525.54	1538.16	3054.33	2579.15	1990.43	1478.81	1065.30	749.92	532.65	336.41	178.72	80.60
CUM. NET PRESENT VALUE (000'$):	-8576	-11988	-13815	-14520	-12697	-11969	-11338	-10765	-10262	-9393	-8756	-8328	-8052	-7879	-7773	-7707	-7671	-7655	-7648
CUM. INTERNAL RATE OF RETURN (%) :					-0.42	-0.30	-0.22	-0.15	-0.11	-0.05	-0.01	0.01	0.02	0.03	0.03	0.03	0.04	0.04	0.04

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 7. BASE CASE WITH A COPPER PRICE OF $6.61/kg

	YEAR																		
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	2453.03	4660.75	6623.18	8338.83	7360.55	7604.39	7751.57	7751.57	7604.39	6132.57	4660.75	3434.24	2453.03	1717.12	1226.51	735.91	367.95	122.65
PROD. REVENUE: (000'$)	0.00	2453.03	4660.75	6623.18	8338.83	7360.55	7604.39	7751.57	7751.57	7604.39	6132.57	4660.75	3434.24	2453.03	1717.12	1226.51	735.91	367.95	122.65
CASH REVENUE: (000'$)	0.00	2044.19	4292.80	6296.11	8052.89	7523.60	7563.75	7727.04	7751.57	7628.92	6377.88	4906.06	3638.66	2616.56	1839.77	1308.28	817.68	429.28	183.98
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.08	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.08	-3631.16	-1803.09	-173.60	6464.46	3517.48	3511.18	3646.44	3670.97	6180.39	5209.71	4018.25	2984.49	2149.30	1512.68	1074.65	677.50	359.19	160.61
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-3631.16	-1803.09	-173.60	6464.46	3517.48	3511.18	3646.44	3670.97	6180.39	5209.71	4018.25	2984.49	2149.30	1512.68	1074.65	677.50	359.19	160.61
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	2585.79	1406.99	1404.47	1458.57	1468.39	2472.16	2083.88	1607.30	1193.79	859.72	605.07	429.86	271.00	143.68	64.25
AFTER TAX CASH FLOW (000'$):	-8576.08	-3631.16	-1803.09	-173.60	3878.68	2110.49	2106.71	2187.86	2202.58	3708.24	3125.82	2410.95	1790.69	1289.58	907.61	644.79	406.50	215.51	96.37
CUM. NET PRESENT VALUE (000'$):	-8576	-11734	-13097	-13211	-10994	-9944	-9033	-8211	-7491	-6437	-5664	-5146	-4811	-4602	-4473	-4394	-4351	-4331	-4323
CUM. INTERNAL RATE OF RETURN (%) :					-0.32	-0.20	-0.12	-0.07	-0.02	0.02	0.05	0.06	0.07	0.08	0.08	0.09	0.09	0.09	0.09

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 8. BASE CASE WITH A COPPER PRICE OF $7.71/kg

	YEAR 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	40	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	40	16
SALES:	0.00	2803.46	5326.58	7569.35	9530.10	8412.06	8690.73	8858.94	8858.94	8690.73	7008.65	5326.58	3924.85	2803.46	1962.42	1401.73	841.04	420.52	140.17
PROD. REVENUE: (000'$)	0.00	2803.46	5326.58	7569.35	9530.10	8412.06	8690.73	8858.94	8858.94	8690.73	7008.65	5326.58	3924.85	2803.46	1962.42	1401.73	841.04	420.52	140.17
CASH REVENUE: (000'$)	0.00	2336.22	4906.06	7195.55	9203.30	8598.40	8644.29	8830.90	8858.94	8718.77	7289.00	5606.92	4158.47	2990.36	2102.60	1495.18	934.49	490.61	210.26
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5208.08	5675.35	6095.89	6469.70	1588.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5208.08	-3339.13	-1189.83	725.85	7614.88	4592.28	4591.72	4750.30	4778.33	7270.24	6120.83	4719.12	3504.29	2523.09	1775.51	1261.55	794.31	420.52	186.90
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-3339.13	-1189.83	725.85	7614.88	4592.28	4591.72	4750.30	4778.33	7270.24	6120.83	4719.12	3504.29	2523.09	1775.51	1261.55	794.31	420.52	186.90
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	290.34	3045.95	1836.91	1836.69	1900.12	1911.33	2908.10	2448.33	1887.65	1401.72	1009.24	710.20	504.62	317.72	168.21	74.76
AFTER TAX CASH FLOW (000'$):	-8576.08	-3339.13	-1189.83	435.51	4568.93	2755.37	2755.03	2850.18	2867.00	4362.14	3672.50	2831.47	2102.58	1513.86	1065.31	756.93	476.58	252.31	112.14
CUM. NET PRESENT VALUE (000'$):	-8576	-11480	-12379	-12093	-9481	-8111	-6920	-5848	-4911	-3671	-2763	-2155	-1762	-1516	-1365	-1272	-1221	-1198	-1189
CUM. INTERNAL RATE OF RETURN (%) :				-0.80	-0.25	-0.13	-0.05	0.00	0.04	0.08	0.10	0.12	0.12	0.13	0.13	0.13	0.13	0.13	0.13

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 9. BASE CASE WITH A COPPER PRICE OF $8.82/kg

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	3153.89	5992.40	8515.51	10721.36	9463.57	9777.07	9966.31	9966.31	9777.07	7884.74	5992.40	4415.45	3153.89	2207.73	1576.95	946.17	473.08	157.69
PROD. REVENUE: (000'$)	0.00	3153.89	5992.40	8515.51	10721.36	9463.57	9777.07	9966.31	9966.31	9777.07	7884.74	5992.40	4415.45	3153.89	2207.73	1576.95	946.17	473.08	157.69
CASH REVENUE: (000'$)	0.00	2620.25	5519.31	8095.00	10353.72	9673.20	9724.82	9934.77	9966.31	9808.61	8200.13	6307.79	4678.20	3364.15	2365.42	1682.08	1051.30	551.93	236.54
OPERATING COSTS (000'$)																			
MINE:	5200.00	5200.00	5200.00	5200.00	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1580.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	5200.00	5675.35	6095.09	6469.70	1580.43	4006.12	4052.57	4080.61	4080.61	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-5200.00	-3047.11	-576.58	1625.29	8765.29	5667.08	5672.25	5854.16	5885.70	8360.08	7031.96	5419.98	4024.10	2896.89	2038.33	1448.44	911.12	481.84	213.18
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-8576.08	-3047.11	-576.58	1625.29	8765.29	5667.08	5672.25	5854.16	5885.70	8360.08	7031.96	5419.98	4024.10	2896.89	2038.33	1448.44	911.12	481.84	213.18
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	650.12	3506.12	2266.83	2268.90	2341.66	2354.28	3344.03	2812.78	2167.99	1609.64	1158.75	815.33	579.38	364.45	192.74	85.27
AFTER TAX CASH FLOW (000'$):	-8576.08	-3047.11	-576.58	975.17	5259.17	3400.25	3403.35	3512.50	3531.42	5016.05	4219.17	3251.99	2414.46	1738.13	1223.00	869.07	546.67	289.10	127.91
CUM. NET PRESENT VALUE (000'$):	-8576	-11226	-11662	-11021	-8014	-6323	-4852	-3531	-2377	-951	92	791	1242	1525	1698	1804	1863	1890	1900
CUM. INTERNAL RATE OF RETURN (%) :				-0.64	-0.18	-0.06	0.02	0.07	0.10	0.13	0.15	0.16	0.17	0.17	0.17	0.18	0.18	0.18	0.18

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 10. BASE CASE WITH A COPPER PRICE OF $9.92/kg

							YEAR												
STATISTICS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Cu in Soln.(t):	0	434	824	1171	1474	1301	1344	1370	1370	1344	1084	824	607	434	304	217	130	65	22
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	434	824	1171	1474	1301	1344	1370	1370	1344	1084	824	607	434	304	217	130	65	22
SALES:	0.00	955.73	1815.00	2580.46	3249.47	2867.10	2962.75	3020.09	3020.09	2962.75	2389.31	1815.00	1338.02	955.73	669.01	477.86	286.72	143.36	47.79
PROD. REVENUE: (000'$)	0.00	955.73	1815.00	2580.46	3249.47	2867.10	2962.75	3020.09	3020.09	2962.75	2389.31	1815.00	1338.02	955.73	669.01	477.86	286.72	143.36	47.79
CASH REVENUE: (000'$)	0.00	796.44	1672.52	2453.03	3137.97	2930.09	2946.82	3010.54	3020.09	2972.31	2484.89	1911.45	1417.66	1019.44	716.79	509.72	318.58	167.25	71.68
OPERATING COSTS (000'$)																			
MINE:	5200.00	5200.00	5200.00	5200.00	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	637.18	1210.65	1720.39	2166.42	1911.55	1975.26	2013.50	2013.50	1975.26	1592.96	1210.65	892.06	637.10	446.03	318.59	191.15	95.58	31.86
TOTAL:	5200.00	5845.27	6418.73	6920.40	2166.42	4515.59	4579.31	4617.54	4617.54	1975.26	1592.96	1210.65	892.06	637.10	446.03	318.59	191.15	95.58	31.86
OPERATING PROFIT :	-5200.00	-5048.83	-4746.21	-4475.45	971.55	-1584.70	-1632.49	-1607.00	-1597.45	997.04	891.93	700.80	525.60	382.26	270.77	191.13	127.42	71.67	39.82
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	2032.70																		
PLANT:	2151.97																		
TOTAL:	4184.67	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-9392.75	-5048.83	-4746.21	-4475.45	971.55	-1584.70	-1632.49	-1607.00	-1597.45	997.04	891.93	700.80	525.60	382.26	270.77	191.13	127.42	71.67	39.82
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	388.62	0.00	0.00	0.00	0.00	398.82	356.77	280.32	210.24	152.90	108.31	76.45	50.97	28.67	15.93
AFTER TAX CASH FLOW (000'$):	-9392.75	-5048.83	-4746.21	-4475.45	582.93	-1584.70	-1632.49	-1607.00	-1597.45	598.22	535.16	420.48	315.36	229.36	162.46	114.68	76.45	43.00	23.89
CUM. NET PRESENT VALUE (000'$):	-9393	-13703	-17372	-20315	-19981	-20769	-21475	-22079	-22601	-22431	-22299	-22209	-22150	-22112	-22089	-22075	-22067	-22063	-22061
CUM. INTERNAL RATE OF RETURN (%) :					-0.89	-0.51	-0.43	-0.36	-0.30	-0.73	-0.50	-0.40	-0.34	-0.31	-0.28	-0.27	-0.26	-0.25	-0.24

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 11. BASE CASE WITH A RECOVERY OF 75%

	YEAR 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	578	1099	1561	1966	1735	1792	1827	1827	1792	1445	1099	809	578	405	289	173	87	29
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	578	1099	1561	1966	1735	1792	1827	1827	1792	1445	1099	809	578	405	289	173	87	29
SALES:	0.00	1274.30	2421.17	3440.61	4332.62	3822.90	3950.33	4026.79	4026.79	3950.33	3185.75	2421.17	1784.02	1274.30	892.01	637.15	382.29	191.15	63.72
PROD. REVENUE: (000'$)	0.00	1274.30	2421.17	3440.61	4332.62	3822.90	3950.33	4026.79	4026.79	3950.33	3185.75	2421.17	1784.02	1274.30	892.01	637.15	382.29	191.15	63.72
CASH REVENUE: (000'$)	0.00	1061.92	2230.03	3270.71	4183.95	3907.86	3929.09	4014.05	4026.79	3963.08	3313.18	2548.60	1890.21	1359.25	955.73	679.63	424.77	223.00	95.57
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	849.58	1614.19	2293.86	2888.56	2548.73	2633.69	2684.66	2684.66	2633.69	2123.94	1614.19	1189.41	849.58	594.70	424.79	254.87	127.44	42.48
TOTAL:	5208.08	6057.66	6822.28	7501.94	2888.56	5152.77	5237.73	5288.70	5288.70	2633.69	2123.94	1614.19	1189.41	849.58	594.70	424.79	254.87	127.44	42.48
OPERATING PROFIT :	-5208.08	-4995.74	-4592.25	-4231.24	1295.39	-1244.92	-1308.63	-1274.66	-1261.91	1329.39	1189.24	934.41	700.81	509.68	361.02	254.84	169.89	95.57	53.09
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	2486.16																		
PLANT:	2632.05																		
TOTAL:	5118.21	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	>>>>>>>>	-4995.74	-4592.25	-4231.24	1295.39	-1244.92	-1308.63	-1274.66	-1261.91	1329.39	1189.24	934.41	700.81	509.68	361.02	254.84	169.89	95.57	53.09
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	518.16	0.00	0.00	0.00	0.00	531.76	475.70	373.76	280.32	203.87	144.41	101.94	67.96	38.23	21.24
AFTER TAX CASH FLOW (000'$):	>>>>>>>>	-4995.74	-4592.25	-4231.24	777.24	-1244.92	-1308.63	-1274.66	-1261.91	797.63	713.54	560.64	420.48	305.81	216.61	152.90	101.94	57.34	31.86
CUM. NET PRESENT VALUE (000'$):	-10326	-14670	-18143	-20925	-20481	-21099	-21665	-22144	-22557	-22330	-22154	-22033	-21955	-21905	-21874	-21856	-21845	-21839	-21837
CUM. INTERNAL RATE OF RETURN (%) :					-0.85	-0.51	-0.43	-0.36	-0.30	-0.61	-0.41	-0.33	-0.29	-0.26	-0.24	-0.23	-0.22	-0.21	-0.21

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 12. BASE CASE WITH A RECOVERY OF 100%

	YEAR																		
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.89	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	3906.06	4256.51	4571.92	4852.28	1191.32	3004.59	3039.43	3060.45	3060.45	1086.40	876.13	665.86	490.63	350.45	245.32	175.23	105.14	52.57	17.52
OPERATING PROFIT :	-3906.06	-3672.46	-3345.40	-3053.39	1109.50	-854.99	-878.36	-852.73	-845.72	1093.30	946.12	735.88	548.99	397.14	280.33	198.57	128.49	70.08	35.04
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-7274.06	-3672.46	-3345.40	-3053.39	1109.50	-854.99	-878.36	-852.73	-845.72	1093.30	946.12	735.88	548.99	397.14	280.33	198.57	128.49	70.08	35.04
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	443.80	0.00	0.00	0.00	0.00	437.32	378.45	294.35	219.59	158.86	112.13	79.43	51.39	28.03	14.02
AFTER TAX CASH FLOW (000'$):	-7274.06	-3672.46	-3345.40	-3053.39	665.70	-854.99	-878.36	-852.73	-845.72	655.98	567.67	441.53	329.39	238.28	168.20	119.14	77.09	42.05	21.03
CUM. NET PRESENT VALUE (000'$):	-7274	-10468	-12997	-15005	-14624	-15049	-15429	-15750	-16026	-15840	-15699	-15604	-15543	-15504	-15480	-15466	-15457	-15453	-15452
CUM. INTERNAL RATE OF RETURN (%) :					-0.82	0.00	0.00	0.00	0.00	-0.57	-0.38	-0.31	-0.27	-0.24	-0.23	-0.21	-0.21	-0.20	-0.20

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 13. BASE CASE WITH A 25% REDUCTION IN TOTAL OPERATING COSTS

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.09	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	2604.04	2837.68	3047.95	3234.85	794.21	2003.06	2026.28	2040.30	2040.30	724.26	584.08	443.90	327.09	233.63	163.54	116.82	70.09	35.05	11.68
OPERATING PROFIT :	-2604.04	-2253.62	-1821.43	-1435.96	1506.61	146.54	134.79	167.42	174.43	1455.43	1238.17	957.83	712.53	513.96	362.11	256.98	163.53	87.61	40.88
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-5972.04	-2253.62	-1821.43	-1435.96	1506.61	146.54	134.79	167.42	174.43	1455.43	1238.17	957.83	712.53	513.96	362.11	256.98	163.53	87.61	40.88
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	602.64	58.62	53.91	66.97	69.77	582.17	495.27	383.13	285.01	205.58	144.84	102.79	65.41	35.04	16.35
AFTER TAX CASH FLOW (000'$):	-5972.04	-2253.62	-1821.43	-1435.96	903.97	87.92	80.87	100.45	104.66	873.26	742.90	574.70	427.52	308.37	217.26	154.19	98.12	52.56	24.53
CUM. NET PRESENT VALUE (000'$):	-5972	-7932	-9309	-10253	-9736	-9693	-9650	-9620	-9586	-9337	-9154	-9030	-8950	-8900	-8870	-8851	-8840	-8835	-8833
CUM. INTERNAL RATE OF RETURN (%) :					-0.65	-0.61	-0.55	-0.47	-0.41	-0.24	-0.18	-0.15	-0.13	-0.11	-0.11	-0.10	-0.10	-0.09	-0.09

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 14. BASE CASE WITH A 50% REDUCTION IN TOTAL OPERATING COSTS

YEAR

STATISTICS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.89	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	1302.02	1418.84	1523.97	1617.43	397.11	1001.53	1013.14	1020.15	1020.15	362.13	292.04	221.95	163.54	116.82	81.77	58.41	35.05	17.52	5.84
OPERATING PROFIT :	-1302.02	-834.78	-297.46	181.46	1903.72	1148.07	1147.93	1187.57	1194.58	1817.56	1530.21	1179.78	876.07	630.77	443.88	315.39	198.58	105.13	46.72
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-4670.02	-834.78	-297.46	181.46	1903.72	1148.07	1147.93	1187.57	1194.58	1817.56	1530.21	1179.78	876.07	630.77	443.88	315.39	198.58	105.13	46.72
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	72.58	761.49	459.23	459.17	475.03	477.83	727.02	612.08	471.91	350.43	252.31	177.55	126.15	79.43	42.05	18.69
AFTER TAX CASH FLOW (000'$):	-4670.02	-834.78	-297.46	108.88	1142.23	688.84	688.76	712.54	716.75	1090.54	918.12	707.87	525.64	378.46	266.33	189.23	119.15	63.08	28.03
CUM. NET PRESENT VALUE (000'$):	-4670	-5396	-5621	-5549	-4896	-4554	-4256	-3988	-3754	-3444	-3217	-3065	-2966	-2905	-2867	-2844	-2831	-2825	-2823
CUM. INTERNAL RATE OF RETURN (%) :				-0.82	-0.35	-0.24	-0.16	-0.10	-0.06	-0.02	0.01	0.02	0.03	0.04	0.04	0.04	0.05	0.05	0.05

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 15. BASE CASE WITH A 75% REDUCTION IN TOTAL OPERATING COSTS

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.89	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	3906.06	3906.06	3906.06	3906.06	0.00	1953.03	1953.03	1953.03	1953.03										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	3906.06	4373.33	4793.87	5167.68	1588.43	3355.11	3401.56	3429.60	3429.60	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-3906.06	-3789.28	-3567.36	-3368.80	712.40	-1205.51	-1240.49	-1221.87	-1214.86	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-7274.06	-3789.28	-3567.36	-3368.80	712.40	-1205.51	-1240.49	-1221.87	-1214.86	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	284.96	0.00	0.00	0.00	0.00	292.47	261.63	205.57	154.18	112.13	79.42	56.06	37.38	21.02	11.68
AFTER TAX CASH FLOW (000'$):	-7274.06	-3789.28	-3567.36	-3368.80	427.44	-1205.51	-1240.49	-1221.87	-1214.86	438.70	392.45	308.35	231.27	168.19	119.14	84.10	56.07	31.54	17.52
CUM. NET PRESENT VALUE (000'$):	-7274	-10569	-13267	-15482	-15237	-15837	-16373	-16832	-17229	-17105	-17008	-16941	-16898	-16871	-16854	-16844	-16838	-16835	-16833
CUM. INTERNAL RATE OF RETURN (%) :		ERROR	ERROR	ERROR	-0.89	-0.24	-0.16	-0.10	-0.06	-0.73	-0.50	-0.40	-0.34	-0.31	-0.29	-0.27	-0.26	-0.25	-0.25

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 16. BASE CASE WITH A 25% REDUCTION IN MINING COSTS

	YEAR																		
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.89	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	2604.04	2604.04	2604.04	2604.04	0.00	1302.02	1302.02	1302.02	1302.02										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	2604.04	3071.31	3491.85	3865.66	1588.43	2704.10	2750.55	2778.58	2778.58	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-2604.04	-2487.25	-2265.33	-2066.78	712.40	-554.50	-589.48	-570.86	-563.85	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-5972.04	-2487.25	-2265.33	-2066.78	712.40	-554.50	-589.48	-570.86	-563.85	731.16	654.08	313.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	284.96	0.00	0.00	0.00	0.00	292.47	261.63	205.57	154.18	112.13	79.42	56.06	37.38	21.02	11.68
AFTER TAX CASH FLOW (000'$):	-5972.04	-2487.25	-2265.33	-2066.78	427.44	-554.50	-589.48	-570.86	-563.85	438.70	392.45	308.35	231.27	168.19	119.14	84.10	56.07	31.54	17.52
CUM. NET PRESENT VALUE (000'$):	-5972	-8135	-9848	-11207	-10962	-11238	-11493	-11707	-11892	-11767	-11670	-11604	-11561	-11533	-11516	-11506	-11500	-11497	-11496
CUM. INTERNAL RATE OF RETURN (%) :					-0.83	-0.24	-0.16	-0.10	-0.06	-0.57	-0.38	-0.31	-0.27	-0.24	-0.23	-0.21	-0.21	-0.20	-0.20

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 17. BASE CASE WITH A 50% REDUCTION IN MINING COSTS

	YEAR 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2382.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.09	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	1302.02	1302.02	1302.02	1302.02	0.00	651.01	651.01	651.01	651.01										
PROCESSING:	0.00	467.27	887.81	1261.62	1500.43	1402.08	1440.53	1476.56	1476.56	1440.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	1302.02	1769.29	2189.83	2563.64	1500.43	2053.09	2099.54	2127.57	2127.57	1440.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-1302.02	-1185.23	-963.31	-764.75	712.40	96.51	61.53	80.15	87.16	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-4670.02	-1185.23	-963.31	-764.75	712.40	96.51	61.53	80.15	87.16	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	284.96	38.60	24.61	32.06	34.86	292.47	261.63	205.57	154.18	112.13	79.42	56.06	37.38	21.02	11.68
AFTER TAX CASH FLOW (000'$):	-4670.02	-1185.23	-963.31	-764.75	427.44	57.91	36.92	48.09	52.30	438.70	392.45	308.35	231.27	168.19	119.14	84.10	56.07	31.54	17.52
CUM. NET PRESENT VALUE (000'$):	-4670	-5701	-6429	-6932	-6688	-6659	-6643	-6625	-6608	-6483	-6386	-6321	-6276	-6249	-6232	-6222	-6216	-6213	-6212
CUM. INTERNAL RATE OF RETURN (%) :					-0.68	-0.62	-0.57	-0.50	-0.44	-0.26	-0.20	-0.17	-0.15	-0.14	-0.13	-0.12	-0.11	-0.11	-0.11

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 18. BASE CASE WITH A 75% REDUCTION IN MINING COSTS

APPENDIX B

CASE 2 STUDY DATA

	YEAR 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	636	1208	1717	2162	1908	1972	2010	2010	1972	1590	1208	890	636	445	318	191	95	32
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	636	1208	1717	2162	1908	1972	2010	2010	1972	1590	1208	890	636	445	318	191	95	32
SALES:	0.00	1401.73	2663.29	3784.67	4765.08	4205.19	4345.37	4429.47	4429.47	4345.37	3504.33	2663.29	1962.42	1401.73	981.21	700.87	420.52	210.26	70.09
PROD. REVENUE: (000'$)	0.00	1401.73	2663.29	3784.67	4765.08	4205.19	4345.37	4429.47	4429.47	4345.37	3504.33	2663.29	1962.42	1401.73	981.21	700.87	420.52	210.26	70.09
CASH REVENUE: (000'$)	0.00	1168.11	2453.03	3597.78	4602.35	4298.64	4322.00	4415.45	4429.47	4359.38	3644.50	2803.46	2079.23	1495.18	1051.30	747.59	467.24	245.30	105.13
OPERATING COSTS (000'$)																			
MINE:	5208.08	5208.08	5208.08	5208.08	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	934.53	1775.61	2523.24	3177.42	2803.60	2897.06	2953.13	2953.13	2897.06	2336.33	1775.61	1308.35	934.53	654.17	467.27	280.36	140.18	46.73
TOTAL:	5208.08	6142.62	6983.70	7731.33	3177.42	5407.64	5501.10	5557.17	5557.17	2897.06	2336.33	1775.61	1308.35	934.53	654.17	467.27	280.36	140.18	46.73
OPERATING PROFIT :	-5208.08	-4974.51	-4530.67	-4133.55	1424.93	-1109.00	-1179.09	-1141.72	-1127.70	1462.33	1308.17	1027.85	770.89	560.65	397.12	280.32	186.88	105.12	58.40
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	2657.69																		
PLANT:	2813.64																		
TOTAL:	5471.33	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	)))))))))	-4974.51	-4530.67	-4133.55	1424.93	-1109.00	-1179.09	-1141.72	-1127.70	1462.33	1308.17	1027.85	770.89	560.65	397.12	280.32	186.88	105.12	58.40
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	569.97	0.00	0.00	0.00	0.00	584.93	523.27	411.14	308.35	224.26	158.85	112.13	74.75	42.05	23.36
AFTER TAX CASH FLOW (000'$):	)))))))))	-4974.51	-4530.67	-4133.55	854.96	-1109.00	-1179.09	-1141.72	-1127.70	877.40	784.90	616.71	462.53	336.39	238.27	168.19	112.13	63.07	35.04
CUM. NET PRESENT VALUE (000'$):	-10679	-15005	-18431	-21149	-20660	-21211	-21721	-22150	-22519	-22270	-22076	-21943	-21857	-21802	-21768	-21747	-21736	-21730	-21727
CUM. INTERNAL RATE OF RETURN (%) :					-0.83	-0.09	-0.02	0.03	0.06	-0.57	-0.38	-0.31	-0.27	-0.24	-0.22	-0.21	-0.20	-0.20	-0.20

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 19. BASE CASE WITH A COPPER GRADE OF 1.8%

	YEAR																		
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	1590	3021	4293	5406	4770	4929	5024	5024	4929	3975	3021	2226	1590	1113	795	477	238	79
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	1590	3021	4293	5406	4770	4929	5024	5024	4929	3975	3021	2226	1590	1113	795	477	238	79
SALES:	0.00	3504.33	6658.22	9461.68	11914.71	10512.98	10863.41	11073.67	11073.67	10863.41	8760.82	6658.22	4906.06	3504.33	2453.03	1752.16	1051.30	525.65	175.22
PROD. REVENUE: (000'$)	0.00	3504.33	6658.22	9461.68	11914.71	10512.98	10863.41	11073.67	11073.67	10863.41	8760.82	6658.22	4906.06	3504.33	2453.03	1752.16	1051.30	525.65	175.22
CASH REVENUE: (000'$)	0.00	2920.27	6132.57	8994.44	11505.87	10746.60	10805.01	11038.63	11073.67	10898.46	9111.25	7008.65	5198.08	3737.95	2628.25	1868.97	1168.11	613.26	262.82
OPERATING COSTS (000'$)																			
MINE:	5200.00	5200.00	5200.00	5200.00	0.00	2604.04	2604.04	2604.04	2604.04										
PROCESSING:	0.00	2336.33	4439.04	6308.10	7943.54	7009.00	7242.64	7382.82	7382.82	7242.64	5840.84	4439.04	3270.87	2336.33	1635.43	1168.17	700.90	350.45	116.82
TOTAL:	5200.00	7544.42	9647.12	11516.19	7943.54	9613.05	9846.68	9986.86	9986.86	7242.64	5840.84	4439.04	3270.87	2336.33	1635.43	1168.17	700.90	350.45	116.82
OPERATING PROFIT :	-5200.00	-4624.15	-3514.55	-2521.75	3562.34	1133.56	958.33	1051.77	1086.81	3655.82	3270.41	2569.62	1927.22	1401.61	992.81	700.81	467.21	262.81	146.01
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	5047.34																		
PLANT:	5343.51																		
TOTAL:	10390.85	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	>>>>>>>>>	-4624.15	-3514.55	-2521.75	3562.34	1133.56	958.33	1051.77	1086.81	3655.82	3270.41	2569.62	1927.22	1401.61	992.81	700.81	467.21	262.81	146.01
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	1424.93	453.42	383.33	420.71	434.73	1462.33	1308.17	1027.85	770.89	560.65	397.12	280.32	186.88	105.12	58.40
AFTER TAX CASH FLOW (000'$):	>>>>>>>>>	-4624.15	-3514.55	-2521.75	2137.40	680.13	575.00	631.06	652.09	2193.49	1962.25	1541.77	1156.33	840.97	595.69	420.48	280.33	157.68	87.60
CUM. NET PRESENT VALUE (000'$):	-15599	-19620	-22277	-23936	-22713	-22375	-22127	-21809	-21676	-21053	-20568	-20236	-20020	-19884	-19799	-19748	-19718	-19703	-19696
CUM. INTERNAL RATE OF RETURN (%) :					-0.61	-0.51	-0.43	-0.36	-0.30	-0.20	-0.15	-0.12	-0.10	-0.09	-0.08	-0.08	-0.07	-0.07	-0.07

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 20. BASE CASE WITH A COPPER GRADE OF 4.5%

	YEAR 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1001	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1001	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	700.87	1331.64	1892.34	2302.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
PROD. REVENUE: (000'$)	0.00	700.87	1331.64	1892.34	2302.52	2103.01	2172.68	2214.73	2214.73	2172.68	1752.16	1331.64	981.21	700.87	490.61	350.43	210.26	105.13	35.04
CASH REVENUE: (000'$)	0.00	584.05	1226.51	1798.88	2300.83	2149.60	2161.07	2207.73	2214.73	2179.69	1822.25	1401.73	1039.62	747.59	525.65	373.79	233.62	122.65	52.56
OPERATING COSTS (000'$)																			
MINE:	2604.04	2604.04	2604.04	2604.04	0.00	1302.02	1302.02	1302.02	1302.02										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	2604.04	3071.31	3491.85	3865.66	1588.43	2704.10	2750.55	2778.58	2778.58	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-2604.04	-2487.25	-2265.33	-2066.78	712.40	-554.50	-589.48	-570.86	-563.85	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-5972.04	-2487.25	-2265.33	-2066.78	712.40	-554.50	-589.48	-570.86	-563.85	731.16	654.08	513.92	385.44	280.32	198.56	140.16	93.44	52.56	29.20
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	284.96	0.00	0.00	0.00	0.00	292.47	261.63	205.57	154.18	112.13	79.42	56.06	37.38	21.02	11.68
AFTER TAX CASH FLOW (000'$):	-5972.04	-2487.25	-2265.33	-2066.78	427.44	-554.50	-589.48	-570.86	-563.85	438.70	392.45	308.35	231.27	168.19	119.14	84.10	56.07	31.54	17.52
CUM. NET PRESENT VALUE (000'$):	-5972	-8135	-9848	-11207	-10962	-11238	-11493	-11707	-11892	-11767	-11670	-11604	-11561	-11533	-11516	-11506	-11500	-11497	-11496
CUM. INTERNAL RATE OF RETURN (%) :					-0.83	-0.24	-0.16	-0.10	-0.06	-0.57	-0.38	-0.31	-0.27	-0.24	-0.23	-0.21	-0.21	-0.20	-0.20

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 21. CASE STUDY II SCHEDULE

							YEAR												
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
STATISTICS																			
Cu in Soln.(t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	318	604	859	1081	954	986	1005	1005	986	795	604	445	318	223	159	95	48	16
SALES:	0.00	1907.76	3624.74	5150.94	6485.23	5724.41	5914.04	6028.51	6028.51	5914.04	4769.39	3624.74	2670.86	1907.76	1335.43	953.89	572.33	286.16	95.39
PROD. REVENUE: (000'$)	0.00	1907.76	3624.74	5150.94	6485.23	5724.41	5914.04	6028.51	6028.51	5914.04	4769.39	3624.74	2670.86	1907.76	1335.43	953.88	572.33	286.16	95.39
CASH REVENUE: (000'$)	0.00	1589.80	3338.57	4896.57	6262.85	5851.21	5882.44	6009.43	6028.51	5933.12	4960.16	3815.51	2829.84	2034.94	1430.82	1017.47	635.92	333.86	143.08
OPERATING COSTS (000'$)																			
MINE:	2604.04	2604.04	2604.04	2604.04	0.00	1302.02	1302.02	1302.02	1302.02										
PROCESSING:	0.00	467.27	887.81	1261.62	1588.43	1402.08	1448.53	1476.56	1476.56	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
TOTAL:	2604.04	3071.31	3491.85	3865.66	1588.43	2704.10	2750.55	2778.58	2778.58	1448.53	1168.17	887.81	654.17	467.27	327.09	233.63	140.18	70.09	23.36
OPERATING PROFIT :	-2604.04	-1481.51	-153.28	1030.91	4674.42	3147.11	3131.89	3230.85	3249.92	4484.59	3792.00	2927.70	2175.66	1567.67	1103.73	783.84	495.74	263.77	119.72
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1636.00																		
PLANT:	1732.00																		
TOTAL:	3368.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-5972.04	-1481.51	-153.28	1030.91	4674.42	3147.11	3131.89	3230.85	3249.92	4484.59	3792.00	2927.70	2175.66	1567.67	1103.73	783.84	495.74	263.77	119.72
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	412.36	1869.77	1258.84	1252.76	1292.34	1299.97	1793.84	1516.80	1171.08	870.27	627.07	441.49	313.53	198.30	105.51	47.89
AFTER TAX CASH FLOW (000'$):	-5972.04	-1481.51	-153.28	618.55	2804.65	1888.27	1879.13	1938.51	1949.95	2690.76	2275.20	1756.62	1305.40	940.60	662.24	470.30	297.44	158.26	71.83
CUM. NET PRESENT VALUE (000'$):	-5972	-7260	-7376	-6970	-5366	-4427	-3615	-2886	-2249	-1484	-921	-544	-300	-147	-53	5	36	51	57
CUM. INTERNAL RATE OF RETURN (%) :				-0.62	-0.20	-0.09	-0.01	0.04	0.07	0.11	0.12	0.14	0.14	0.15	0.15	0.15	0.15	0.15	0.15

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 22. CASE STUDY II WITH A COPPER PRICE OF $6/kg

APPENDIX C

CASE 3 STUDY DATA

											YEAR								
STATISTICS	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19
Cu in Soln.(t):	0	633	1203	1709	2153	1899	1963	2001	2001	1963	1583	1203	886	633	443	317	190	95	32
Recirculated(t):	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Produced Cu (t):	0	633	1203	1709	2153	1899	1963	2001	2001	1963	1583	1203	886	633	443	317	190	95	32
SALES:	0.00	1395.36	2651.18	3767.47	4744.22	4186.08	4325.61	4409.34	4409.34	4325.61	3488.40	2651.18	1953.50	1395.36	976.75	697.68	418.61	209.30	69.77
PROD. REVENUE: (000'$)	0.00	1395.36	2651.18	3767.47	4744.22	4186.08	4325.61	4409.34	4409.34	4325.61	3488.40	2651.18	1953.50	1395.36	976.75	697.68	418.61	209.30	69.77
CASH REVENUE: (000'$)	0.00	1162.80	2441.88	3581.42	4581.43	4279.10	4302.36	4395.38	4409.34	4339.57	3627.93	2790.72	2069.78	1488.38	1046.52	744.19	465.12	244.19	104.65
OPERATING COSTS (000'$)																			
MINE:	2737.37	2737.37	2737.37	2737.37	0.00	1368.68	1368.68	1368.68	1368.68										
PROCESSING:	0.00	758.18	1440.55	2047.09	2577.82	2274.55	2350.37	2395.86	2395.86	2350.37	1895.46	1440.55	1061.46	758.18	530.73	379.09	227.45	113.73	37.91
TOTAL:	2737.37	3495.55	4177.92	4784.46	2577.82	3643.23	3719.05	3764.54	3764.54	2350.37	1895.46	1440.55	1061.46	758.18	530.73	379.09	227.45	113.73	37.91
OPERATING PROFIT :	-2737.37	-2332.75	-1736.04	-1293.04	2003.61	635.87	583.31	630.84	644.79	1989.20	1732.48	1350.17	1008.33	730.20	515.79	365.10	237.66	130.46	66.74
CAPITAL COSTS (000'$)																			
MINE:																			
LEACH:	1864.00																		
PLANT:	1732.00																		
TOTAL:	3596.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
BEFORE TAX CASH FLOW (000'$):	-6333.37	-2332.75	-1736.04	-1203.04	2003.61	635.87	583.31	630.84	644.79	1989.20	1732.48	1350.17	1008.33	730.20	515.79	365.10	237.66	130.46	66.74
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	801.44	254.35	233.32	252.34	257.92	795.68	692.99	540.07	403.33	292.08	206.32	146.04	95.07	52.18	26.70
AFTER TAX CASH FLOW (000'$):	-6333.37	-2332.75	-1736.04	-1203.04	1202.16	381.52	349.98	378.50	386.88	1193.52	1039.49	810.10	605.00	438.12	309.47	219.06	142.60	78.28	40.05
CUM. NET PRESENT VALUE (000'$):	-6333	-8362	-9675	-10466	-9778	-9589	-9437	-9295	-9168	-8829	-8572	-8398	-8285	-8214	-8170	-8143	-8128	-8121	-8117
CUM. INTERNAL RATE OF RETURN (%) :					-0.58	-0.47	-0.39	-0.32	-0.26	-0.17	-0.12	-0.09	-0.08	-0.07	-0.06	-0.05	-0.05	-0.05	-0.05

NOTE: Processing includes smelting, refining, transportation, leaching, cementation, and a smelter penalty of 11 kg/tonne.

TABLE 23 . CASE STUDY III SCHEDULE

APPENDIX D

CONVENTIONAL MINING DATA

	YEAR					
	1	2	3	4	5	6
STATISTICS						
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
PROD. REVENUE: (000'$)	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
CASH REVENUE: (000'$)	6808.75	8170.50	8170.50	8170.50	8170.50	9532.25
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	13821.80	13821.80	13821.80	13821.80	13821.80	13821.80
OPERATING PROFIT :	-7013.05	-5651.30	-5651.30	-5651.30	-5651.30	-4289.55
BEFORE TAX CASH FLOW (000'$):	-7013.05	-5651.30	-5651.30	-5651.30	-5651.30	-4289.55
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	0.00
AFTER TAX CASH FLOW (000'$):	-7013.05	-5651.30	-5651.30	-5651.30	-5651.30	-4289.55
CUM. NET PRESENT VALUE (000'$):	-7013	-11927	-16200	-19916	-23147	-25280
CUM. INTERNAL RATE OF RETURN (%) :						

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 24. CONVENTIONAL MINING CASH FLOW SCHEDULE

	YEAR 1	2	3	4	5	6
STATISTICS						
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	4085.25	4085.25	4085.25	4085.25	4085.25	4085.25
PROD. REVENUE: (000'$)	4085.25	4085.25	4085.25	4085.25	4085.25	4085.25
CASH REVENUE: (000'$)	3404.37	4085.25	4085.25	4085.25	4085.25	4766.12
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	13821.80	13821.80	13821.80	13821.80	13821.80	13821.80
OPERATING PROFIT :	>>>>>>>>	-9736.55	-9736.55	-9736.55	-9736.55	-9055.67
BEFORE TAX CASH FLOW (000'$):	>>>>>>>>	-9736.55	-9736.55	-9736.55	-9736.55	-9055.67
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	0.00
AFTER TAX CASH FLOW (000'$):	>>>>>>>>	-9736.55	-9736.55	-9736.55	-9736.55	-9055.67
CUM. NET PRESENT VALUE (000'$):	-10417	-18884	-26246	-32648	-38215	-42717
CUM. INTERNAL RATE OF RETURN (%) :						

NOTE: Processing includes milling, smelting, a smelter penalty of 11 kg/tonne, refining, and transportation.

TABLE 25. CONVENTIONAL MINING CASE WITH COPPER SELLING FOR $1.10/kg

	YEAR					
STATISTICS	1	2	3	4	5	6
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	12255.74	12255.74	12255.74	12255.74	12255.74	12255.74
PROD. REVENUE: (000'$)	12255.74	12255.74	12255.74	12255.74	12255.74	12255.74
CASH REVENUE: (000'$)	10213.12	12255.74	12255.74	12255.74	12255.74	14298.37
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	13821.80	13821.80	13821.80	13821.80	13821.80	13821.80
OPERATING PROFIT :	-3608.68	-1566.05	-1566.05	-1566.05	-1566.05	476.57
BEFORE TAX CASH FLOW (000'$):	-3608.68	-1566.05	-1566.05	-1566.05	-1566.05	476.57
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	190.63
AFTER TAX CASH FLOW (000'$):	-3608.68	-1566.05	-1566.05	-1566.05	-1566.05	285.94
CUM. NET PRESENT VALUE (000'$):	-3609	-4970	-6155	-7184	-8080	-7938
CUM. INTERNAL RATE OF RETURN (%) :						-0.85

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 26. CONVENTIONAL MINING CASE WITH COPPER SELLING FOR $3.31/kg

	YEAR					
	1	2	3	4	5	6
STATISTICS						
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	16340.99	16340.99	16340.99	16340.99	16340.99	16340.99
PROD. REVENUE: (000'$)	16340.99	16340.99	16340.99	16340.99	16340.99	16340.99
CASH REVENUE: (000'$)	13617.49	16340.99	16340.99	16340.99	16340.99	19064.49
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	13821.80	13821.80	13821.80	13821.80	13821.80	13821.80
OPERATING PROFIT :	-204.30	2519.20	2519.20	2519.20	2519.20	5242.69
BEFORE TAX CASH FLOW (000'$):	-204.30	2519.20	2519.20	2519.20	2519.20	5242.69
TAXES PAYABLE @ 40% (000'$):	0.00	1007.68	1007.68	1007.68	1007.68	2097.08
AFTER TAX CASH FLOW (000'$):	-204.30	1511.52	1511.52	1511.52	1511.52	3145.62
CUM. NET PRESENT VALUE (000'$):	-204	1110	2253	3247	4111	5675
CUM. INTERNAL RATE OF RETURN (%) :		6.40	7.29	7.39	7.40	7.40

NOTE: Processing includes milling, smelting, a smelter penalty of 11 kg/tonne, refining, and transportation.

TABLE 27. CONVENTIONAL MINING CASE WITH COPPER SELLING FOR $4.41/kg

	YEAR					
STATISTICS	1	2	3	4	5	6
Cu (t):	5560.68	5560.68	5560.68	5560.68	5560.68	5560.68
SALES:	12255.74	12255.74	12255.74	12255.74	12255.74	12255.74
PROD. REVENUE: (000'$)	12255.74	12255.74	12255.74	12255.74	12255.74	12255.74
CASH REVENUE: (000'$)	10213.12	12255.74	12255.74	12255.74	12255.74	14298.37
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	6943.68	6943.68	6943.68	6943.68	6943.68	6943.68
TOTAL:	15270.37	15270.37	15270.37	15270.37	15270.37	15270.37
OPERATING PROFIT :	-5057.25	-3014.62	-3014.62	-3014.62	-3014.62	-972.00
BEFORE TAX CASH FLOW (000'$):	-5057.25	-3014.62	-3014.62	-3014.62	-3014.62	-972.00
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	0.00
AFTER TAX CASH FLOW (000'$):	-5057.25	-3014.62	-3014.62	-3014.62	-3014.62	-972.00
CUM. NET PRESENT VALUE (000'$):	-5057	-7679	-9958	-11940	-13664	-14147
CUM. INTERNAL RATE OF RETURN (%) :						

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 28. CONVENTIONAL MINING CASE WITH A COPPER GRADE OF 1.35%

	YEAR					
STATISTICS	1	2	3	4	5	6
Cu (t):	7414.24	7414.24	7414.24	7414.24	7414.24	7414.24
SALES:	16340.99	16340.99	16340.99	16340.99	16340.99	16340.99
PROD. REVENUE: (000'$)	16340.99	16340.99	16340.99	16340.99	16340.99	16340.99
CASH REVENUE: (000'$)	13617.49	16340.99	16340.99	16340.99	16340.99	19064.49
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	8392.25	8392.25	8392.25	8392.25	8392.25	8392.25
TOTAL:	16718.94	16718.94	16718.94	16718.94	16718.94	16718.94
OPERATING PROFIT :	-3101.45	-377.95	-377.95	-377.95	-377.95	2345.55
BEFORE TAX CASH FLOW (000'$):	-3101.45	-377.95	-377.95	-377.95	-377.95	2345.55
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	938.22
AFTER TAX CASH FLOW (000'$):	-3101.45	-377.95	-377.95	-377.95	-377.95	1407.33
CUM. NET PRESENT VALUE (000'$):	-3101	-3430	-3716	-3964	-4180	-3481
CUM. INTERNAL RATE OF RETURN (%) :						-0.27

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 29 . CONVENTIONAL MINING CASE WITH A COPPER GRADE OF 1.8%

	YEAR					
STATISTICS	1	2	3	4	5	6
Cu (t):	9267.80	9267.80	9267.80	9267.80	9267.80	9267.80
SALES:	20426.24	20426.24	20426.24	20426.24	20426.24	20426.24
PROD. REVENUE: (000'$)	20426.24	20426.24	20426.24	20426.24	20426.24	20426.24
CASH REVENUE: (000'$)	17021.87	20426.24	20426.24	20426.24	20426.24	23830.61
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	9840.82	9840.82	9840.82	9840.82	9840.82	9840.82
TOTAL:	18167.51	18167.51	18167.51	18167.51	18167.51	18167.51
OPERATING PROFIT :	-1145.64	2258.73	2258.73	2258.73	2258.73	5663.10
BEFORE TAX CASH FLOW (000'$):	-1145.64	2258.73	2258.73	2258.73	2258.73	5663.10
TAXES PAYABLE @ 40% (000'$):	0.00	903.49	903.49	903.49	903.49	2265.24
AFTER TAX CASH FLOW (000'$):	-1145.64	1355.24	1355.24	1355.24	1355.24	3397.86
CUM. NET PRESENT VALUE (000'$):	-1146	33	1058	1949	2724	4413
CUM. INTERNAL RATE OF RETURN (%) :		0.18	0.83	1.04	1.12	1.20

NOTE: Processing includes milling, smelting, a smelter penalty of 11 kg/tonne, refining, and transportation.

TABLE 30. CONVENTIONAL MINING CASE WITH A COPPER GRADE OF 2.25%

	YEAR					
	1	2	3	4	5	6
STATISTICS						
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
PROD. REVENUE: (000'$)	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
CASH REVENUE: (000'$)	6808.75	8170.50	8170.50	8170.50	8170.50	9532.25
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	10366.35	10366.35	10366.35	10366.35	10366.35	10366.35
OPERATING PROFIT :	-3557.60	-2195.85	-2195.85	-2195.85	-2195.85	-834.10
BEFORE TAX CASH FLOW (000'$):	-3557.60	-2195.85	-2195.85	-2195.85	-2195.85	-834.10
TAXES PAYABLE @ 40% (000'$):	0.00	0.00	0.00	0.00	0.00	0.00
AFTER TAX CASH FLOW (000'$):	-3557.60	-2195.85	-2195.85	-2195.85	-2195.85	-834.10
CUM. NET PRESENT VALUE (000'$):	-3558	-5467	-7127	-8571	-9827	-10241
CUM. INTERNAL RATE OF RETURN (%) :						

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 31 . CONVENTIONAL MINING CASE WITH TOTAL
OPERATING COST RECOVERED BY 25%

	YEAR					
STATISTICS	1	2	3	4	5	6
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
PROD. REVENUE: (000'$)	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
CASH REVENUE: (000'$)	6808.75	8170.50	8170.50	8170.50	8170.50	9532.25
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	6910.90	6910.90	6910.90	6910.90	6910.90	6910.90
OPERATING PROFIT :	-102.15	1259.60	1259.60	1259.60	1259.60	2621.35
BEFORE TAX CASH FLOW (000'$):	-102.15	1259.60	1259.60	1259.60	1259.60	2621.35
TAXES PAYABLE @ 40% (000'$):	0.00	503.84	503.84	503.84	503.84	1048.54
AFTER TAX CASH FLOW (000'$):	-102.15	755.76	755.76	755.76	755.76	1572.81
CUM. NET PRESENT VALUE (000'$):	-102	555	1126	1623	2056	2837
CUM. INTERNAL RATE OF RETURN (%) :		6.40	7.29	7.39	7.40	7.40

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 32. CONVENTIONAL MINING CASE WITH TOTAL OPERATING COSTS RECOVERED BY 50%

	YEAR					
	1	2	3	4	5	6
STATISTICS						
Cu (t):	3707.12	3707.12	3707.12	3707.12	3707.12	3707.12
SALES:	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
PROD. REVENUE: (000'$)	8170.50	8170.50	8170.50	8170.50	8170.50	8170.50
CASH REVENUE: (000'$)	6808.75	8170.50	8170.50	8170.50	8170.50	9532.25
OPERATING COSTS (000'$)						
MINE:	8326.69	8326.69	8326.69	8326.69	8326.69	8326.69
PROCESSING:	5495.11	5495.11	5495.11	5495.11	5495.11	5495.11
TOTAL:	3455.45	3455.45	3455.45	3455.45	3455.45	3455.45
OPERATING PROFIT :	3353.30	4715.05	4715.05	4715.05	4715.05	6076.80
BEFORE TAX CASH FLOW (000'$):	3353.30	4715.05	4715.05	4715.05	4715.05	6076.80
TAXES PAYABLE @ 40% (000'$):	1341.32	1886.02	1886.02	1886.02	1886.02	2430.72
AFTER TAX CASH FLOW (000'$):	2011.98	2829.03	2829.03	2829.03	2829.03	3646.08
CUM. NET PRESENT VALUE (000'$):	2012	4472	6611	8471	10089	11902
CUM. INTERNAL RATE OF RETURN (%) :						

NOTE: Processing includes milling, smelting,
a smelter penalty of 11 kg/tonne,
refining, and transportation.

TABLE 33. CONVENTIONAL MINING CASE WITH TOTAL OPERATING COSTS RECOVERED BY 75%